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THE MORRO VELHO MINE, BRAZIL.

(From a Photograph.)

THE

MINING AND METALLURGY

OF

GOLD AND SILVER.

13360.S

ODD FELLOWS' LIBRARY ASSOCIATION.
SAN FRANCISCO, CAL.

BY

J. ARTHUR PHILLIPS,

MINING ENGINEER.

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Dedication.

TO

HENRY HUSSEY VIVIAN, Esq., M.P.,

THIS WORK,

TREATING OF SOME OF THE BRANCHES OF AN ART
WHICH HIS FAMILY HAS FOR A LONG PERIOD SUCCESSFULLY PRACTISED,
AND WHICH
HE HAS HIMSELF SO GREATLY CONTRIBUTED TO ADVANCE,

IS RESPECTFULLY DEDICATED

BY

THE AUTHOR.

ODD FELLOWS' LIBRARY ASSOCIATION. SAN FRANCISCO, CAL.

PREFACE.

THIS treatise has been undertaken with the hope that it may, to some extent, supply a want in our technical literature, which is very deficient in books treating of the mining and metallurgy of the precious metals.

In preparing this work for publication, care has been taken to obtain data directly from the best and most authentic sources. The connexion of the author, as a Mining Engineer, with various large metallurgical establishments, together with his repeated visits to, and residence in, some of the principal mining districts of both Europe and America, have afforded him many opportunities for observation and investigation.

His object has rather been to render this volume practically useful as a record of well-authenticated facts, and of the results of actual experience, than to advance new theories, or to accumulate additional evidence in support of old ones. He has, therefore, contented himself with merely stating the results both of his own observations, and those of others; or at most with suggesting the nature of the forces producing the effects described. He believes that our knowledge of chemical geology is not, as yet, sufficiently advanced to warrant an attempt to form a general theory of the formation of mineral veins.

Recent observations and experience appear, however, to lead to three important conclusions. First, that the most productive gold-bearing rocks are by no means exclusively confined to the Silurian period; secondly, that aqueous agencies have been, and still are, actively at work in the formation of mineral deposits; and, thirdly, that gold ledges are not more liable than ordinary metalliferous veins to become impoverished in depth.

The general order adopted has been, first to describe, in each instance, the principal gold and silver producing districts, and to give such statistical information as could be obtained respecting their yield and importance. Then follows an account of the methods employed for extracting the ores; and, lastly, a description of the apparatus made use of for their mechanical and metallurgical treatment.

With regard to the yield of silver from the various districts producing ores of that metal, it may be remarked that, in a few instances, it has been found necessary to depart from the official estimates; since ores raised in one country have been metallurgically treated in another, and the resulting metal has consequently figured in the returns of both.

The information derived either from other books or from private sources, has, in most instances, been duly acknowledged; but the Author takes this opportunity of returning thanks to his friends for many valuable suggestions of too general a character to admit of specific mention in the body of the work. He is under especial obligations to the Messrs. Taylor, of London; Mr. T. F. Cronise and Mr. P. N. McKay, of San Francisco; Mr. W. Watt, of Grass Valley; and to the gentlemen of the State Geological Survey of California. He also takes this opportunity of expressing his thanks to the principal mill owners and mining engineers of California and Nevada; without the facilities which these gentlemen have so kindly afforded, the Author would have been unable to give the very large amount of facts relating to the mines of the Pacific Coast which their liberality has enabled him to collect.

The table and formulæ on page 236 have been contributed by Mr. C. W. Merrifield, F.R.S., to whom, and to Mr. F. W. Rudler, the author begs to return his thanks for assistance rendered whilst the book was passing through the press.

KENSINGTON,
July, 1867.

CONTENTS.

GOLD.

CHAPTER I.

MODE OF OCCURRENCE, AND GEOLOGICAL POSITION.

PAGE

Analyses of Native Gold—Age of Auriferous Rocks—Gold of the Jurassic Period—Quartz Veins of recent Formation—Gold Veins not impoverished in Depth—Alluvial Deposits 1

CHAPTER II.

GOLD REGIONS OF THE OLD WORLD.

Great Britain and Ireland—France—Spain and Portugal—Italy—Switzerland—Holland—Russian Possessions—Germany—Austrian Empire—Central and Southern Asia—China and Japan—Africa 15

CHAPTER III.

UNITED STATES OF AMERICA—ATLANTIC OR APPALACHIAN GOLD FIELDS.

Discovery of Gold in the Southern States—Virginia—North Carolina—South Carolina—Georgia—Tennessee and Alabama—Description of Appalachian Gold Fields—Principal Gold Mines of the Southern States. 29

CHAPTER IV.

UNITED STATES OF AMERICA—GOLD FIELDS OF THE PACIFIC COAST.

Discovery of Gold in California—Situation and Extent of Auriferous Districts—Shallow and Deep Placers—Quartz Veins—Mariposa—Tuolumne—Calaveras—Amador—El Dorado—Placer—Nevada and Sierra Counties, &c.—Statistics of Gold produced in the United States 37

CHAPTER V.

MEXICO, CENTRAL AMERICA, AND SOUTH AMERICA.

PAGE

Mexico pre-eminently a Silver-producing Country—Nicaragua—Chontales Mining Company—Brazil—St. John d'El Rey Company—Imperial Brazilian—Don Pedro North d'El Rey—Rossa Grande Gold Mining Company—East d'El Rey—New Granada—Mariquita and New Granada Mining Company—Peru and Bolivia 77

CHAPTER VI.

BRITISH POSSESSIONS—NORTH AMERICAN COLONIES.

Discovery of Gold in Nova Scotia—Geology of Gold Region—Corrugated Quartz at Waverly—Impracticable Mining Laws—Statistics of Gold obtained—Canadian Gold Fields—Principal Gold Washings in Canada—Report of Parliamentary Committee—Discovery of Gold in British Columbia—Extent of the Gold Fields—Gold chiefly obtained from Alluvial Diggings 91

CHAPTER VII.

BRITISH POSSESSIONS—AUSTRALIA AND NEW ZEALAND.

Discovery of Gold in Australia—Victoria the most productive Division of the Australian Continent—Geology of the Gold-bearing Rocks of Victoria—Quartz Veins—Auriferous Pliocene Gravels—Primitive Mining—Quartz Veins productive at all Depths yet attained—Port Phillip and Colonial Gold Mining Company—Yield of some of the Principal Mines—Total Amount of Gold exported from Victoria—Gold exported from New South Wales—South Australia and Tasmania—Queensland—Discovery of Gold in New Zealand—Principal Gold Fields of that Country—Yield of Gold—Annual Production of the Gold Mines of the World 101

CHAPTER VIII.

GOLD WASHING IN CALIFORNIA AND AUSTRALIA.

Placer Mining—The Pan—Rocker—Long Tom—Puddling Box—Sluice—River Mining—Beach Mining—Water Supply—Eureka Canal—Hydraulic Mining—Amount of Water required—Cost of Water, and Method of Measurement—Dry Washing 128

CHAPTER IX.

VEIN MINING IN CALIFORNIA AND AUSTRALIA.

PAGE

Quartz Mining—Testing Gold Quartz—Batea—Horn Spoon—Arrastre—Chilian Mill—Stamping Mill—Amalgamating in Battery—Iron Battery Box—Screens—Single Cams—Separation of Gold—Blankets—Amalgamated Copper Plates—Cleaning up—Settling Pits—Amalgamator—Attwood's System of Amalgamation—Loss of Gold—Concentration of Tailings—Rocker—Concave Buddle—Bradford's Separator—Extraction of Gold from Sulphides—Baux and Guiod's Amalgamator—Chlorination Process—Analyses of Californian Pyrites—Retorting—Melting—Tabular Statement of the Operations of the principal Californian Quartz Mills 165

CHAPTER X.

VEIN MINING IN HUNGARY AND BRAZIL—SODIUM AMALGAM.

Auriferous Veins at Schemnitz—Stamping Mills—Hungarian Bowls—Shaking Tables—Concentration of Auriferous Slimes—Goldlütte—Scheidetrog—Results obtained—Mode of Occurrence of Gold at Morro Velho—Stamping—Strakes—Treatment of First Tailings—Treatment of Second Tailings—Amalgamation of concentrated Ores—Loss of Gold and Mercury—Cost of Working—Application of Sodium Amalgam—Not generally employed in California—Advantages stated to result from its Use 203

CHAPTER XI.

ASSAY OF AURIFEROUS ORES—ESTIMATION OF GOLD CONTAINED IN QUARTZ—REFINING—ASSAY OF GOLD BULLION.

Assay of Gold Quartz—Fusion with Litharge or Red Lead—Auriferous Pyrites—Cupellation—Inquartation—Parting—Assay Table—Table showing Proportion of Gold in Auriferous Quartz—Refining—Platinum Vessels—Cast Iron Pans—Assay of Bullion—Quantity of Lead necessary for Cupellation of Alloys of Gold and Copper—Assay Laboratory—The Touchstone 226

SILVER.

CHAPTER XII.

MODE OF OCCURRENCE AND GEOLOGICAL POSITION.

PAGE

Native Silver—Antimonial Silver—Native Amalgam—Principal Silver Ores—Sulphides—Arsenides—Selenides—Tellurides—Chloride—Iodide—Bromide—Carbonate—Silicate—Chief Sources of Silver—Geological Position 247

CHAPTER XIII.

PRINCIPAL SILVER MINES OF THE OLD WORLD.

Silver Mines of the United Kingdom—Norway—Sweden—Transylvania and the Banat—Saxony and Bohemia—Mines of the Hartz—Silver Mines of the Alps—France—Spain—Altai Mountains—Daouria 252

CHAPTER XIV.

SILVER MINES OF NORTH AMERICA.

Principal Mining Districts of Mexico—Production of Silver in Mexico up to 1845—Produce of Ores treated—Guanaxuato—Zacatecas—Fresnillo—Real del Monte—History and Operations of the Real del Monte Mining Company—Principal Mining Districts of Nevada—Comstock Vein—Discovery of Silver at Virginia—Yield during Four Years—Sutro Tunnel—Reese River District—Systems of Veins—Amador District—Ravenswood—Cortez—Washington—Union—Smoky Valley—Twin River—San Antonio—Other Mining Districts—Table showing Number of Reduction Works, &c. 266

CHAPTER XV.

CENTRAL AMERICA AND SOUTH AMERICA.

Guatemala—Mines of the Central American Mining Company—Mining District of Alotepec—Produce of Silver—South America—Mines of Peru—Annual Production of Silver—Bolivia—Mines of Potosi—Their Production of Silver—Chili—Mines of Coquimbo—Annual Production of Chili—New Granada—Santa Ana Mines—Table showing Production of Silver in various Countries . 311

CHAPTER XVI.

TREATMENT OF SILVER ORES BY AMALGAMATION—PATIO PROCESS, &c.

PAGE

Patio Process—Magistral—Salt—Mercury—Lime—Copper Precipitate—Composition of Ores—Rough Stamping—Fine Grinding—Raspar—Loss of Gold—The Patio—Incorporar—Treading of Torta—Washing—Straining Amalgam—Distillation—Loss of Silver—Loss of Mercury—Chemical Reactions of the Patio—Results of the Patio Process—At Guanaxuato—Fresnillo—Real del Monte—Virginia City—Amalgamation by Hot Process—Estufa Amalgamation 321

CHAPTER XVII.

TREATMENT OF SILVER ORES BY AMALGAMATION—BARREL PROCESS.

Freiberg—Barrel Amalgamation—When introduced—Composition of Ores—Chlorination—Amalgamation—Distillation of Amalgam—Refining—Constante—Grinding Crude Ores—Calcining with Salt—Sifting and Grinding—Amalgamation at Constante—Treatment of Amalgam—Melting and Refining—Treatment of Residues—Cost of treating Ores at Constante—Cost at Real del Monte—The Barrel Process in Nevada—Method of conducting, and Cost of Operation 364

CHAPTER XVIII.

TREATMENT OF SILVER ORES BY AMALGAMATION—PAN PROCESS.

Stamping—Common Pan—Varney's Pan—Wheeler's Pan—Hepburn and Peterson's Pan—Separators—Working in Pans—Retorting—Arrangement of Reduction Works—Treatment of roasted Ores in Pans 390

CHAPTER XIX.

TREATMENT OF ARGENTIFEROUS ORES AND PRODUCTS BY SOLUTION AND PRECIPITATION.

Augustin's Process—Roasting with Salt—Lixiviation and Precipitation—Ziervogel's Process—Roasting—Solution of Sulphate of Silver—Precipitation of Silver by Copper—Von Patera's Process—Roasting with Common Salt—Solution of Chloride of Silver in Hyposulphite of Soda—Precipitation by Sulphide of Sodium—Treatment of Sulphide of Silver 409

CHAPTER XX.

CONCENTRATION OF THE PRECIOUS METALS IN METALLIC LEAD—SMELTING.

Preliminary Observations—Smelting Auriferous Silver Ores for Matts—Extraction of the Precious Metals in Bath of fused Lead—Concentration of Silver in Matts, and Fusion with roasted Lead Ores, without Addition of Iron—Reduction of unroasted Lead Ores by Metallic Iron—Castillian Furnace—Reduction of partially roasted Lead Ores by Metallic Iron—Results obtained at Wildberg 426

CHAPTER XXI.

EXTRACTION OF THE PRECIOUS METALS FROM ARGENTIFEROUS AND AURIFEROUS LEAD OBTAINED BY SMELTING.

PAGE

Continental Process of Cupellation—Refining Silver obtained from Cupel—English System of treating Argentiferous Lead—Calcination—Concentration of Silver by Crystallisation—Pattinson's Process—French Process—Refining enriched Lead—Costs and Results of Desilverising—Liquation 448

CHAPTER XXII.

TREATMENT OF ARGENTIFEROUS GALENA AT PONTGIBAUD.

Preparation of Lits de Grillage—Roasting—Preparation of Lits de Fusion—Smelting in Castillian Furnace—Improving or Calcining—Crystallising—Refining—Reducing—Re-smelting Rich Slags—Roasting Matts—Treatment of Calcined Dross—Treatment of Lead Cinder—Treatment of Lead Fume—Losses of Lead and Silver—Summary of Costs 468

CHAPTER XXIII.

SMELTING SILVER ORES IN MEXICO.

Roasting—Fusion—Cupellation—Vaso—Galeme—Ñufla 497

CHAPTER XXIV.

ASSAY OF SILVER ORES AND BULLION.

Argentiferous Galena—Assay of Silver Ores Proper—Scorification—Cupellation—Assay Table—Assay of Silver Bullion 504

Appendix 517

Index 523

LIST OF WOODCUTS.

FIG. PAGE
The Morro Velho Mine, Brazil . . . . . . *Frontispiece.*
1. Diggings at the Soimanofsk Mines 23
2. Section at Maine Boy's Tunnel 44
3. Tunnels under Table Mountain 45
4. Corrugated Quartz, Laidlaw's Farm 93
5. Clunes, North Cross Section 114
6. ,, South ,, ... 115
7. Washing with Pan 133
8. Section of Cradle 134
9. Washing with Cradle 134
10. Section of Tom and Riffle Box 138
11. Plan ,, ,, ,, ... 138
12. Plan of Sluice Box 141
13. Section ,, ... 141
14. Zig-zag Riffles 145
15. Under-current Sluice 147
16. Tail Sluices, Yuba River 148
17. Flume near Smartsville 154
18. Pressure Box, Yuba River 156
19. Hydraulic Mining, Washing down Bank 157
20. Sluice and Tunnel, Timbuctoo 159
21. Iron Pipes 162
22. Batea 167
23. Horn, showing Method of Cutting Spoon 168
24. Arrastre 169
25. Stamp Grate, with Round Holes 175
26. ,, ,, Oblong ,, ... 175
27. Transverse Section of Iron Battery Box 176
28. Elevation of Iron Battery Box 176
29. Single Cam 177
30. Section of Blanket Board 180
31. Section of Washing Tank 182
32. Section of Amalgamator 185
33. Plan ,, ... 185
34. Section of Concave Buddle 192
35. Plan ,, ,, ... 193
36. Baux and Guiod's Amalgamator 196
37. Melting and Assay Furnace 229
38. Assay Laboratory 243
39. Section of Comstock Vein 291

FIG. PAGE
40. Patio at the Gould and Curry . . . 335
41. Patio at Guanaxuato . . . 341
42. Washing Apparatus employed at Guanaxuato . . . 342
43. Manga or Strainer . . . 347
44. Mould containing Brick of Amalgam . . . 347
45. Lifting the Capellina . . . 348
46. Capellina . . . 349
47. Sectional Elevation of Amalgamating Barrels . . . 368
48. Plan of Amalgamating Barrels . . . 368
49. Vertical Section of Revolving Calciner . . . 374
50. Horizontal ,, ,, ,, . . . 375
51. Horizontal Mills . . . 378
52. Vertical Section of Common Pan . . . 393
53. Plan of Common Pan . . . 394
54. Wheeler's Pan . . . 398
55. Hepburn and Peterson's Pan . . . 399
56. Front Elevation of Retort . . . 405
57. Longitudinal Section of Retort . . . 406
58. Lixiviating Tub . . . 411
59. Ziervogel's Process, Arrangement of Apparatus . . . 416
60. Von Patera's Process, ,, ,, . . . 421
61. Clausthal Furnace, Vertical Section . . . 439
62. ,, ,, Section through Tuyere . . . 439
63. ,, ,, Horizontal Section through Tuyeres . . . 439
64. Elevation of Castillian Furnace . . . 443
65. Vertical Section ,, . . . 443
66. Horizontal Section ,, . . . 443
67. Vertical Section of Cupelling Furnace . . . 449
68. Horizontal Section ,, . . . 449
69. Plan of Crystallising Pots . . . 457
70. Elevation of ,, . . . 457
71. ,, Pot and Crane . . . 458
72. ,, Refinery . . . 462
73. Horizontal Section of Refinery . . . 462
74. Vertical ,, ,, . . . 463
75. Elevation of Roasting Furnace, Pontgibaud . . . 472
76. Horizontal Section ,, ,, . . . 473
77. Vertical ,, ,, ,, . . . 473
78. Elevation of Castillian Furnace ,, . . . 477
79. Side View of Slag Waggon ,, . . . 479
80. End ,, ,, ,, . . . 479
81. Plan of ,, ,, . . . 479
82. Improving Furnace—
Elevation ,, . . . 482
83. Horizontal Section ,, . . . 483
84. Section through Tap Hole ,, . . . 483

REFERENCE TO PLATES, ETC.

I. Map of the Grass Valley Mining District, California . . *To follow Page* 62
II. Ten-Stamp Battery . . . . " 178
III. System of Blankets and Riffles employed at Grass Valley, &c. . . . . " 184
IV. Attwood's Amalgamator . . . . " 188
V. Machinery employed for the Extraction of Gold at Schemnitz, Lower Hungary . . . . " 210
VI. Amalgamating Machinery, Morro Velho Mine, Brazil . " 220
VII. Varney's Amalgamating Pan . . . . " 396
VIII. Elevation of Silver Reduction Works, Auburn, Nevada . " 408

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GOLD.

CHAPTER I.

MODE OF OCCURRENCE AND GEOLOGICAL POSITION.

ANALYSES OF NATIVE GOLD—AGE OF AURIFEROUS ROCKS—GOLD OF THE JURASSIC PERIOD—QUARTZ VEINS OF RECENT FORMATION—GOLD VEINS NOT IMPOVERISHED IN DEPTH—ALLUVIAL DEPOSITS.

MODE OF OCCURRENCE.—This metal occurs in the following forms:—

ALLOYS.

Native Gold.—An Alloy of Gold and Silver, associated with small quantities of copper, iron, and other metals.
Palladium Gold.—Gold and Palladium—Porpezite.
Rhodium Gold.—Gold and Rhodium.

AMALGAM.

Gold Amalgam.—A native Amalgam of Gold and Mercury.

ORES.

Sylvanite, or Graphic Tellurium.—Telluride of Gold and Silver.
Nagyagite.—Telluride of Lead containing Gold, Silver, and Copper.

Of the foregoing combinations, Native Gold is the only one of great commercial importance, as furnishing nearly the whole of this metal annually obtained from the different gold-producing districts of the world. Palladium gold occurs to some extent in the mines of Gongo Soco in Brazil, and a small quantity of graphic tellurium is found in those of Transylvania. The other native alloys of gold can only be regarded in the light of mineralogical curiosities.

Gold is usually found, in alluvial washings, in the shape of fine particles and water-worn plates and scales, but crystallised specimens are occasionally met with. These crystals, which are usually small, are generally in the form of octahedrons, although Shepard describes a pentagonal dodecahedron, from California, two-fifths of an inch in diameter: an octahedral crystal from the same country, described by F. Alger, had a diameter of eight-tenths of an inch.

Native gold, occurring in veins, is most frequently found in a quartzose gangue, in which it is often associated with iron and copper pyrites, arsenical pyrites, blende and galena. Gold and iron pyrites are very intimately associated, although the gold appears always to exist in the metallic state, since in almost every gold-producing district, when sulphide of iron has become decomposed, by weathering, into hydrated oxide of iron, gold becomes apparent, and is readily separated by washing. When this metal is encased in undecomposed copper pyrites, iron pyrites, or any other sulphide, or arsenide, its separation, either by washing or amalgamation, is attended with considerable difficulty, although the whole of the gold may be readily extracted by smelting the concentrated pyrites, either with lead ore, or with litharge.

The gold found in ordinary gold quartz, in addition to being disseminated in a more or less finely divided state in the associated sulphides, presents itself in threads, thin plates, and grains of varying dimensions. These are not unfrequently apparent to the naked eye, but rock showing no traces of visible gold is often sufficiently rich to yield a large profit after deducting working expenses.

Native gold invariably contains a certain amount of silver, and almost always traces of copper and iron. The silver associated with native gold is not combined with it according to the laws of atomic proportion, but forms with it an alloy in which silver represents from one-hundredth to more than one-half the total weight of the mixture.

The gold of Australia averages from 960 to 966 thousandths fine, whilst that of California contains from 875 to 885 thousandths of pure gold.

The composition of native gold from various localities is given in the following table:—

TABLE

Showing the Composition of various Samples of Native Gold.

Locality.	Analyst.	Gold.	Silver.	Copper.	Iron.
Russian Empire—					
Petropawlowsk washings . .	G. Rose	86·81	13·19	Trace	Trace.
Boruschkoi "	"	83·85	16·15	"	"
Zarewo-Nicolajewsk, near Miask, washings	"	89·35	10·65	"	"
Berezovsk Mine, in brown Hematite	"	93·78	5·94	0·08	0·04
Berezovsk, in quartz	"	91·88	8·03	0·09	Trace.
Transylvania—					
Sta. Barbara Mine, at Füses, scales in porphyry with quartz	"	84·89	14·68	0·04	0·13
Vöröspatak, in porphyry with quartz	"	60·49	38·74	0·77	...
Australia—					
South Australia	A. S. Thomas	87·78	6·07	...	6·15
Bathurst	J. H. Henry	95·69	3·92	...	0·16
Ballarat	F. Claudet	99·25	0·65	...	...
New Zealand—					
Locality not known	"	96·25	3·55	...	...
Wales—					
Vigra and Clogau	"	88·50	5·00	...	
Welsh Gold Mining Co.	"	76·40	22·70	...	...
British Columbia—					
Locality not given	"	86·10	13·50	...	...
Stephens Creek	"	79·50	19·70	...	...
Cariboo	"	84·25	14·90	...	...
South America—					
Marmato	Boussingault	73·45	26·48	...	...
Antioquia, New Granada, washings	"	64·93	35·07	...	...
Brazil	D'Arcet	94·00	5·85	...	...
United States and Canada—					
Feather River, California, scales	Rivot	89·10	10·50	...	0·20
American Fork, California, scales	"	90·90	8·70	...	0·20
Mariposa Estate, Quartz Gold.	F. Claudet	81·00	18·70	...	...
Georgia	W. W. Mather	95·579	4·421	Trace	Trace.
Rivière du Loup, Canada . .	T. S. Hunt	86·40	13·60	"	"

The larger pieces of water-worn gold are called by the miners *nuggets;* but these seldom exceed a few pounds in weight, and are generally accompanied by more or less of the quartzose gangue forming the original veinstone.

One of the finest lumps of native gold now in existence, is that preserved in the collection of the Russian School of Mines, which weighs about 97 lbs.

The largest piece of gold ever found, was probably the great Australian nugget known as the "Sarah Sands," which weighed 233 lbs. 4 oz. troy.

GEOLOGICAL POSITION OF GOLD.—The general characteristics of the gold-bearing rocks throughout the world are exceedingly alike, since, whatever their original structure or composition may have been, they have become so assimilated by metamorphic action as to exhibit a very striking resemblance to each other.

They generally consist of slaty deposits, frequently talcose, although sometimes chloritic or argillaceous, and in these the gold-bearing quartz, which is the almost universal matrix of this metal, is generally found most productive. Valuable gold veins are also occasionally found in granite, gneiss and syenite, but these do not so often form the enclosing rock as the metamorphosed shales and slates above referred to.

Veins of auriferous quartz have most frequently the same strike and dip as the shales in which they are enclosed, but in some instances they not only intersect the shales at varying angles, but have also a separate and independent dip. When rocks remain stratified, in nearly the same position in which they were originally deposited, they are rarely found to be highly auriferous; but when, on the contrary, they have been invaded by eruptive masses, are broken up, or raised on edge, and have assumed a crystalline texture, there is good reason to anticipate the presence of the precious metal.

Generally speaking, it may be said that a considerable proportion of the gold-bearing rocks of the world belong to the two lowest geological groups, the azoic and palæozoic, or rather to the latter. For although we are not aware that rocks proved to be azoic have been found to be auriferous, yet it is often impossible to distinguish between these two groups, since in localities where the formation is metalliferous, the strata have generally so changed their original character, as to render it exceedingly difficult to recognise their exact position in the geological series.*

The metamorphosed auriferous strata of the Ural Mountains have been shown by Sir Roderick Murchison to be palæozoic, and the Australian rocks, associated with veins of auriferous quartz, have also afforded abundant evidence of Silurian origin. Mr. Selwyn, the director of the Colonial Geological Survey, has fully satisfied himself that the gold-bearing veins of Victoria are confined to rocks of the palæozoic age, chiefly to those belonging to the Lower Silurian division, ascending from the Llandeilo to the Upper Caradoc formation inclusive. Arguing from his own experience in the Ural Mountains and elsewhere, as well as from facts collected by other observers in the various gold-producing countries of the world, Sir R. Murchison became impressed with the conviction that all the more productive auriferous rocks belong to the palæozoic period, and he considers this question so fully settled, that he remarks, "My chief article of belief has now proved to be true, namely, that the rocks which are most auriferous are of Silurian age."† It would appear, however, that the gold-producing rocks of California and Nevada present a remarkable exception to this general rule, since Professor Whitney, the State Geologist of California, has collected numerous fossils of undoubted Jurassic origin, found *in situ*, in the immediate proximity of one of the largest and best defined quartz veins of the Pacific coast. When describing the geology of the district in the vicinity of the Pine Tree vein, which is one of the largest comprehended within the limits of the Mariposa Estate, he remarks:‡ "Within the slate formation are interstratified sandstones, which are, in some cases, very distinctly bedded, as for instance on the west slope of Hell Hollow. The beds are often several feet in thickness, and like the slates themselves, in all stages of alteration, as is made evident by the differences of hardness, colour and texture which they exhibit. The region of least metamorphism seems to have been from the Pine Tree mine, west, to the slope of Juniper Ridge, and southward, to near the village of Bear Valley. It was in this region that Mr. King found *in situ* fossils, by which the age of this formation was clearly made out to be Jurassic."*

---

* The Discovery by Sir William Logan of fossil remains in the Lower Laurentian rocks of Canada, renders it extremely doubtful whether the term *azoic* can be correctly applied, even to the oldest formations known to geologists.

† Siluria, p. 474.

‡ Geological Survey of California, p. 226.

The period of the formation of auriferous veins is less readily determined than the age of the enclosing rocks, although they may be presumed to have often originated at the time of the metamorphic action, by which the change in the strata themselves was effected. It is, however, tolerably evident that this action has not been confined to any particular geological epoch, and it would appear that these changes, which are probably exceedingly slow in their progress, may have been repeated at periods of time very far removed from one another.

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* The following description of fossils from the auriferous slates of California, is by F. B. Meek; see Appendix, Vol. I. Geological Survey of California, p. 477. "The fossils from the auriferous slates, collected on the Mariposa Estate, having been by mistake sent for investigation in part to Mr. Gabb, and in part to the writer, were regarded by both of us, independently, as most probably of Jurassic age. This opinion was expressed by Mr. Gabb in a communication to the California Academy of Sciences, in November 1864 (see Proceed, Vol. III. p. 172), and by myself at about the same time in a communication to the State Geologist. In his paper alluded to above, Mr. Gabb described the following species, viz., *Lima Erringtoni*, *Pholadomya orbiculata*, and *Belemnites Pacificus*. He likewise mentioned a *Pecten*, and a *Nucula* or *Leda*. Two of these species were also described by the writer, under other names, in a communication prepared for publication, but fortunately not published.

"As it was considered by the State Geologist desirable that I should examine all of the few fossils yet found in these slates, they were placed in my hands for investigation; and drawings and descriptions have been prepared for publication.

"In examining the specimens formerly sent, much difficulty was experienced in determining the generic relations of two or three bivalves resembling an oblique *Inoceramus*, and varying considerably from each other in form and convexity, but agreeing exactly in surface markings. Being all casts, more or less compressed and otherwise distorted, it was difficult to understand how they could belong to one species, as their surface markings and some other points of resemblance seemed to indicate, though this was suspected to be the case. Amongst the collections subsequently sent on, and now under examination, these same forms are observed. One is an obliquely oval rather convex shell, with a very prominent terminal strongly incurved beak, while the other is much more compressed, with a straighter, dorsal outline, and a beak scarcely distinct from the cardinal margin. On examining the collections more recently received it was soon observed that the compressed specimens with an inconspicuous beak, are all right valves, while all of those with a prominent strongly incurved beak are left valves.

"This fact, together with their other characters, leave little room for doubting that these are the opposite valves of one or two species of the genus *Aucella*. As this genus is, so far as known, entirely confined to the Jurassic rocks, while an *Amussium*-like shell from the same slates is closely allied to a Jurassic species, and the genus *Belemnites* is not generally regarded as dating back beyond the commencement of the Jurassic period, I can scarcely entertain a doubt that these gold-bearing slates really belong to that epoch, and probably to some of its lower members, at which horizon most of the known European species of *Aucella* are said to occur."

The impregnation with gold of the rocks of the Ural has been shown by Murchison to have taken place at a comparatively recent date, but in many of the other important gold regions we have not sufficient data to enable us to fix, with any degree of exactitude, the epoch at which the concentration of this metal into veins took place. In California, however, this could not have been effected prior to the deposition of the Jurassic strata in which they occur. A further evidence of the occasional recent formation of quartz veins is derived from the fact, that in one of the detrital beds near Volcano, in Amador County, a distinctly marked quartz vein is observed to cut through the beds of sand and gravel, and presents unmistakeable evidence of having been formed subsequently to their deposition, by the action of water holding silica in solution. This vein is chiefly composed of calcedony and agate, but portions of it are more or less stained by a ferruginous deposit. This is by no means a solitary case, many other localities having been noticed, where quartz veins, almost identical in their general features with those met with in the auriferous slates, must have been formed during the most recent geological epochs.*

Auriferous veins, like all others, vary exceedingly, in not only their dimensions, but also in their productiveness. It is, however, generally observed that the widest veins are not usually the richest, and that some of the laminæ running parallel with the enclosing walls are uniformly more productive than others. It therefore not unfrequently happens that a portion of a vein, sufficiently rich to enable it to be treated with advantage, is separated from another band, comparatively barren, by a distinct heading, or false wall. As a general rule those veins are most productive which afford considerable quantities of disseminated sulphides; although, near the surface, these have, in almost every instance, become decomposed, thereby liberating the enclosed granular gold, and staining the quartz of a brown or reddish colour. When gold occurs in a vein of hard white quartz without sulphides, it is in most instances found in flakes and granules of considerable size, and is consequently visible to the naked eye; but such veins, although sometimes affording fine cabinet specimens, are not often regularly and remuneratively productive. Some of the most steadily remunerative veins, on the other hand, are only of moderate size, and seldom exhibit visible gold, and this is particularly noticeable in those which, like the Norambagua lead in Grass Valley, California, are divided by numerous thin seams of slate into bands of various thickness. In such veins the gold is commonly in a finely divided state, and principally occurs in the parallel headings, marking the lines of deposition of the quartz. It was formerly believed that veins of auriferous quartz become gradually less productive as greater depths from the surface are attained, but more extensive experience would tend to show that this is in reality not the case. Gold mines which have for many years been continuously worked in various part of the world, have fluctuated considerably in their richness at different depths, but it has not been found that these variations in any way correspond with a gradual impoverishment in the deeper levels. In a communication addressed to Sir R. Murchison, who inclines to the opinion that gold-bearing veins generally diminish in value as they descend in depth, Mr. Selwyn remarks as follows: "There is undoubtedly good evidence that those upper portions of the quartz veins, which have been naturally removed by denudation, and now form the gold drifts, were often far richer than any we now find at the surface; but in drawing conclusions from this evidence, we should not forget that in all probability many hundreds of vertical feet of quartz veins have been thus naturally broken up, crushed and washed, and the fact of the veins, so abraded, being still frequently very rich on their present surface, goes far, I think, to prove that the diminution of yield in depth, even though admitted to be true, on a large scale, is still so slow as not to be appreciable within any depth to which ordinary mining operations are carried."* Mr. Selwyn concludes by expressing an opinion, "that the extraction of gold from quartz reefs, if properly conducted, may be regarded as an occupation, which will prove as permanently profitable in Victoria, as tin and copper mining have been in Great Britain."

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* Geological Survey of California, p. 276.

In California, the early quartz miners were also fully impressed with the idea that the outcrops of the leads were more productive of gold than the deeper portions of the same veins, and as soon as the quartz extracted ceased to afford remunerative returns, they usually suspended operations, without extending their explorations to any considerable depth. Within the last few years, however, their opinions in this respect have become materially changed, since the workings of the deeper mines would lead to the conclusion, that, although leads of gold-bearing quartz, like all other metalliferous veins, vary materially in their yield in different portions of their extent, both in length and depth, there is no evidence to indicate a progressive falling off in their yield in the deeper workings. The North Star, Allison Ranch, and Eureka veins, in Grass Valley, will, among many others that might be selected, serve to illustrate the fact, that the Californian mines do not become sensibly impoverished in depth, as, in common with all the mines of this district, they are at the present time quite as productive as they have ever been at any period since the commencement of operations.

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* Siluria, p. 496.

The North Star vein is now worked on its inclination to a depth of 750 feet, and affords quartz yielding, on an average, gold of the value of 7*l.* per ton of 2,000 lbs., whereas in the upper levels the gross value of the gold extracted did not exceed 4*l.* per ton.* The Allison Ranch Mine has now reached the depth of 500 feet from the surface, and, during the first three months of 1866, yielded a net profit of above 20,000*l.*

The Eureka Mine is being sunk to the 400 ft. level, and produces quartz at this depth fully equal to the average of that raised during any former period, having during the last year yielded 12,200 tons of vein-stuff, affording an average of above 9*l.* 12*s.* per ton. Hayward's Mine, in Amador County, is another still more striking instance of the produce of a vein of quartz not decreasing as it goes down. This ledge is worked on its inclination to a depth of above 1,250 feet, and yields quartz of much greater value than that obtained from the same vein at shallower levels. On taking into consideration the whole of the circumstances of the case, it is by no means remarkable that at first the opinion should have become prevalent that quartz veins in most instances become impoverished in depth.

It will be readily admitted that metalliferous veins are exceedingly variable in their yield at different depths, and it may be supposed that those only which showed evidences at the surface of being more or less auriferous would be at first operated on. These, after having been worked to a greater or less depth, will, in accordance with the general law, begin to show signs of having become less auriferous; and although a further prosecution of the operations would probably have led to fresh discoveries, the miners, in a country where capital is not readily obtained, and where wages are high, become discouraged, and finally transfer their operations to other outcrops, presenting a sufficient amount of gold to render its extraction profitable. Another reason for the former prevalence of this impression may be traced to the fact, that gold is almost universally associated with a greater or less amount of iron pyrites and other sulphides, and these, becoming oxidised at shallow depths, liberate the enclosed gold, which is thus readily collected by amalgamation, although the deeper, and consequently less decomposed portions of the vein, which may in reality have been equally auriferous, afforded to the early miners less satisfactory results. With the improved methods of treatment, however, which have now come into general operation, this difficulty has to a great extent disappeared, and as all the auriferous sulphides are being at the present time carefully collected for subsequent elaboration, the average production of a vein has generally been found to be sustained at all depths to which the miner has hitherto penetrated.*

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* The ton of 2,000 lbs. is the standard of weight almost universally adopted in the United States of America.

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* The moulds of cubical crystals of iron pyrites are frequently found in the quartz of auriferous veins, and more particularly so near the surface, thus showing that the formation of the pyrites must have been as old as that of the vein itself. In such cases, although the iron has often been entirely removed by chemical action, the cavities left sometimes contain finely divided gold, obviously liberated by the decomposition of pyrites. The gold contained in crystallised pyrites enclosed in quartz, is readily rendered apparent by placing the specimen, for a few hours, in a warm place, in nitric acid, by which the pyrites is dissolved, and finely powdered, or filiform, gold will partially occupy the resulting cavities. With regard to the age of auriferous quartz veins, it has been already shown that many of them must evidently be of comparatively recent date, but in some cases the deposition of gold-bearing quartz would appear to be taking place even at the present time. At Steamboat Springs, near Virginia, in the State of Nevada, and in other localities on the Pacific coast, numerous parallel deposits of quartz, assuming the form of veins, are taking place along a line of boiling springs now in a state of great activity. The quartz from this locality exactly resembles that of the ordinary auriferous quartz veins of California, and besides small quantities of iron and copper pyrites, contains oxide of iron and traces of manganese. On making an examination of this quartz for gold and silver, we were unable to find an appreciable quantity of either of these metals; but Mr. Laur, who made a similar investigation of this quartz, succeeded in finding specimens containing small quantities of gold. (Annales des Mines, Sixième Série, p. 421.) These facts would, therefore, not only tend to lead to the conclusion that auriferous veins are under certain conditions deposited from silicious solutions, but also to explain the action by which many of the slates of the auriferous period may have become metamorphosed and silicified.

We are indebted to Dr. Oxland, formerly manager of the Works belonging to the

In addition to the gold occurring in veins, considerable quantities are sometimes found forming an integral component of the rocks themselves, as in the syenites of Bogoslofsk in the Ural, and some of the schists in Siberia, &c.* M. Laur† has also discovered the

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Borax Lake Company, Lake County, California, for the following note on the occurrence of gold and silver in that locality:—

"In the Sulphur Bank at Borax Lake, sulphur is constantly in course of formation, with the evolution of aqueous vapour, carbonic acid, and boracic acid, but without any sulphuretted hydrogen, which might have been expected to be present. The smell of carbonic acid is remarkably pungent. The gaseous matters issuing from the *Soffioni* in gentle blowers are usually at the temperature of about 95° Fahr. They appear to be the agency by which gold, silver, mercury, and iron are brought up from below and deposited in cavities near the surface. Sulphur is deposited on the sides of the cavities, either in groups of crystals, or in highly translucent amorphous masses of a beautiful light lemon-yellow colour. Sometimes the sulphur is intermixed with cinnabar, but more frequently with very fine crystals of iron pyrites, and with pulverulent silica in masses blackened by some hydro-carbon which is difficult to isolate. The iron pyrites may be separated by dissolving off the sulphur with bisulphide of carbon, and washing off the silica with water. It is found associated with silver and a trace of gold.

"On the sides of the cavities of the blowers, gelatinous silica is sometimes found coating opalised silica in varying degrees of induration, according to its depth from the surface, presenting examples of opal or hydrated silica in its various stages of formation, from gelatinous silica up to the hardest opal. The indurated silica is sometimes colourless, but is more frequently permeated with cinnabar or iron pyrites, and blackened by the tarry matter before alluded to. Sometimes from a diffusion of cinnabar throughout the mass, in minute quantity, it is delicately tinted of a pinkish colour. The cinnabar is also found in striæ, and occasionally even in veins and concretionary masses of some thickness. Where the bituminous matter occurs in the largest quantity, and the mass is quite black and friable, cinnabar is replaced by metallic mercury.

"In another locality of similar character, about ten miles distant, gold has been found with cinnabar in crystalline masses of some size. In the same place, a vein of apparently compact quartz, about ten inches in thickness, was found to be so friable that it could be easily taken out with the hand in small conchoidal fragments, most of which rapidly fell into fine powder. From its great resemblance to a vein occurring in the Mexican Mine, Virginia City, which is many feet in thickness, and contains $20 to $30 of gold and silver to the ton, attention was drawn to it, and it proved, on being assayed, to contain silver, with a trace of gold, to the value of $15 per ton.

"These phenomena present indubitable evidences of the volatility of gold, silver, mercury, and iron, in presence of aqueous vapour associated with sulphuretted hydrogen, carbonic acid, and boracic acid. Whether the contemporaneous association of these substances may produce a definite compound possessing peculiar powers of solution and volatilization under the influence of elevated temperature, although probable, yet remains to be proved."

* Siluria, p. 481.

† Annales des Mines, Sixième Série, p. 434.

presence of gold in the metamorphic shales of Mariposa County, California; and similar deposits, containing a considerable amount of the precious metal, are known to occur near Lincoln in that State.

It is not, however, from the treatment of auriferous quartz that the principal portion of the gold of commerce is procured, a very large proportion of it, probably more than two-thirds, being obtained from alluvial diggings, in which the gold is separated from the more or less superficial detritus by washing. In some of these deposits nature has been for ages performing the operations of crushing and concentrating on a vast scale, and has deposited the precious metal in positions from which it can be obtained without a large expenditure of either labour or capital. To this circumstance are attributable the great variations which are observable in the production of gold throughout the world; since, on the discovery of new and extensive alluvial diggings, an almost unlimited supply of unskilled labour can at once be applied to its extraction, whilst the same weight of metal, if retaining its original position, could only be collected by the application of a large amount of capital and skilled labour. Indeed, experience clearly shows, that had not this natural concentration taken place, the larger proportion of the gold annually brought into the market could not have been profitably collected, since, in the majority of cases, the veins, from which it must have been originally derived, are not sufficiently rich to defray the expenses of raising and crushing.

It does not seem to admit of a doubt that these deposits have been principally derived from the degradation or breaking up of auriferous veins, and the distribution of the detritus thus formed, chiefly by the agency of running water. It further appears to be conclusively proved, that this gold-bearing drift is, at least, of two distinct geological epochs, both comparatively modern, although the latter period is distinctly separated from the earlier, and its materials chiefly derived from the disintegration and redistribution of the older placers. In California these appear to be distinctly referable to a river-system different from that which now exists, flowing at a higher level, or over a less elevated continental mass, but sometimes in the direction of the main valleys of the present period.

The sources to which the chief annual supply of gold is referable are, therefore, the following:—

1st. Auriferous veins, most frequently enclosed in metamorphic slates.

2nd. The distribution of placer gold by ancient river-systems, known as *Deep diggings.*

3rd. The distribution of placer gold by the present river-system, giving rise to the *Shallow diggings.*

The separation of gold from its original matrix, and its concentration and deposition among and beneath strata of gravel, sand and clay, are the result of causes acting through immense periods of time, and which, although they have not yet ceased, are probably much less energetic than they were at a former but not very remote, geological epoch. The strata constituting the earth's crust are constantly undergoing abrasion and decomposition from various meteorological causes, of which one of the most active and powerful is the alternate freezing and thawing of water retained in the fissures and crevices of stratified rocks, which tend to disintegrate and wear away the more elevated and exposed portions, and to carry down the abraded and loosened fragments and spread them out over the surface of the country at lower levels. On the declivities of lofty and rugged mountain chains, where torrents, either from the fall of rain or the rapid melting of the snows, are of frequent occurrence, the streams, become suddenly swollen, rush with violence down their flanks into the valleys beneath, and thus develop a force capable of rapidly eroding the rocks over which they pass. In addition to this mechanical force, chemical action frequently lends its aid to effect the disintegration of auriferous rocks. In proportion as the gold-bearing strata are worn away, the sulphides with which they are constantly associated become oxidised, and, being thus disintegrated, the rocks are themselves reduced to fragments, which being carried by the action of water into the valleys, the metallic particles, as the heaviest, are first deposited, and sink to the bottom, whilst the lighter earthy and silicious portions are removed by the current to a greater distance.

Geologists are not yet agreed with regard to the identity in origin of the older auriferous sands and gravels, and the modern alluvial formations affording gold: Sir R. Murchison, in particular, strongly insisting that the more or less superficial deposits, constituting the deep diggings, are in no way to be confounded with the modern drifts resulting from present atmospheric causes, but rather that they are the result of diluvial currents closely connected with great physical changes in the earth's surface, such as the elevation of some of the principal mountain chains, &c. The vast accumulation of this *débris*, in some of the mining districts of California and Siberia, would indeed lead to the conclusion that the forces at present in operation were totally inadequate to produce such stupendous results; but until a much more thorough examination shall have been made of some of the great gold-producing districts, it would be injudicious to attempt to arrive at any definite conclusion on this subject.

Baron Richthofen is, however, at the present time engaged in an examination of the deep placers and ancient river-courses of California, and there can be no doubt but that a large amount of additional information on this subject will be derived from the investigations of this indefatigable explorer and able geologist.

CHAPTER II.

GOLD REGIONS OF THE OLD WORLD.

GREAT BRITAIN AND IRELAND—FRANCE—SPAIN AND PORTUGAL—ITALY—SWITZERLAND—HOLLAND—RUSSIAN POSSESSIONS—GERMANY—AUSTRIAN EMPIRE—CENTRAL AND SOUTHERN ASIA—CHINA AND JAPAN—AFRICA.

HAVING made certain general observations on the occurrence of gold, we will now proceed to a consideration of some of the principal auriferous districts of the world.

GREAT BRITAIN AND IRELAND.—*Cornwall and Devon.*—The tin streams of Cornwall have been long known to afford occasional specimens of gold, but not in sufficient quantities to make its collection a matter of any commercial importance. Carew says—"Tynners doe also find little hoppes of gold amongst their Owre, which they keepe in quils and sell to the Goldsmithes, oftentimes with little better gaine than *Glaucus* exchange."* Pryce mentions a piece of gold, found in Cornwall, weighing 15 dwt. 16 gr.† Many of the copper gossans are also known to contain minute quantities of gold, but we are not aware that it has, in any instance, been extracted with advantage. At the Britannia and Poltimore mines, near North Molton, in Devonshire, gold has also been found in small quantities, but it never paid the cost of extraction.

Wales.—In North Wales, especially in Merionethshire, the older slaty rocks have long been known to be more or less auriferous. The gold-bearing district of this country would appear to be chiefly confined to an area of about twenty-five square miles, principally lying on the north of the turnpike road leading from Dolgelly to Barmouth. In this region the Cambrian rocks are overlaid by the Silurian, and the general geological features of the country strongly resemble those of some other auriferous localities. Among the veins which have attracted the most attention are those of the Dol-y-frwynog, Prince of Wales, and Vigra and Clogau mines, of which the latter only is understood to have been ever worked with remunerative results. So long ago as 1844, a paper was read before the British Association by Mr. Arthur Dean, who stated that a complete system of auriferous veins existed throughout the whole of the Snowdonian or Lower Silurian formation of North Wales. In consequence of this statement operations were commenced at Cwn Eisen, but the results obtained not having been found satisfactory, they were finally abandoned. Machinery for crushing and amalgamation was, about two years afterwards, erected at Dol-y-frwynog, but after operating on several hundred tons of quartz, the result was in this instance also a failure. Of all the auriferous veins that have been worked in the neighbourhood of Dolgelly, that of the Vigra and Clogau has certainly been the most productive. This mine is situated at a height of about a thousand feet above the level of the sea, the workings being prosecuted on what is called the St. David's or Gold Lode. This lode, which is almost vertical, runs nearly east and west, and is composed of quartz, more or less impregnated with sulphides of iron, lead, and copper. The veinstone also, at one period, particularly during the year 1862, exhibited for a short distance a considerable amount of disseminated gold. This rich deposit is at present reported to have been entirely worked out; but for a considerable time rather large amounts of gold were returned from the property.

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* Carew's Survey of Cornwall, 1602, book i.

† Pryce's Mineralogia Cornubiensis, p. 52.

The following table, for which we are indebted to Mr. R. Hunt, keeper of Mining Records, shows the returns made by the various Welsh gold mines from 1860 to 1864, or during the time of their largest yield:—

GOLD PRODUCED IN MERIONETHSHIRE.

	1861.		1862.		1863.		1864.	
	Gold. oz.	Value. £	Gold. oz.	Value £	Gold. oz.	Value. £	Gold. oz.	Value. £
Vigra and Clogau .	2,866	10,816	5,299	20,390	526	1,674	2,381	8,434
Cefn Coch . . .	—	—	—	—	25	73	346	970
Castell Carn Dochan	—	—	—	—	—	—	141	394
Prince of Wales .	—	—	—	—	—	—	63	166
Gwyn-fynydd . .	—	—	—	—	—	—	6	17

The number of tons of quartz crushed in Wales during the year 1865 is returned as having been 4,270, yielding 1,664 oz. of gold, or about 0.39 oz. per ton.

The total quantities of gold raised from the commencement of operations in the North Wales Gold District, up to April 1st, 1866, are estimated as follows:—

	Gold obtained. Oz.
Old Dol-y-frwynog	117
Prince of Wales	63
Cwn Eisen	176
Gwyn-fynydd	6
Cefn Coch	478
Castell Carn Dochan	182
Vigra and Clogau	11,778
Total	12,800

As above stated, the productive portion of the Vigra and Clogau lode appears to have become exhausted, and although expensive and efficient machinery has been erected on a large scale, with a view to the treatment of the average quality of quartz produced from the vein, the results hitherto obtained have been far from satisfactory; and, unless some marked improvement should ere long take place, it is to be feared that the Vigra and Clogau may ultimately add another to the list of Welsh gold mines that have absorbed a much larger amount of this metal than they have ever produced.* The old Dol-y-frwynog was reworked for gold in 1864, but without any commercial success, since the average yield of the quartz obtained was only about 26 grains per ton.

Scotland.—Gold has also been found at Lead Hills in Scotland, several hundred men having been employed there in washing gold sands during the reign of James V. Pennant says—"In the reign of *James IV.* the *Scots* did separate gold from the sand by washing. In the following the *Germans* found gold there which afforded the king great sums; three hundred men were employed for several summers and about 100,000*l.* sterling procured." †

Ireland.—In the county of Wicklow, a considerable amount of excitement was caused, in 1796, by the discovery of gold in the *débris* of the valley, at the base of Croghan-Kinshela. These diggings were carried on for about two years, when the works were destroyed by the Irish rebels; but although a considerable quantity of gold was extracted, one specimen in particular weighing no less than twenty-two ounces, the general results obtained do not appear to have been very satisfactory. In 1801 the operations were resumed with a view of discovering the gold-bearing veins; but after the expenditure of large sums, without success, in various mining operations, the locality was again abandoned.

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* Some further discoveries of gold at Vigra and Clogau have been recently announced.

† Pennant's Scotland, vol. ii. p. 130.

France possesses no known valuable gold mines, but the sands of some of her rivers are, to a certain extent, auriferous. The only quartz vein which has been found to contain gold, is that of La Gardette in the Department of the Isère, which is from two to three feet in width, and enclosed in gneiss. Gold was discovered in this locality in 1700, and workings were intermittently carried on, up to 1841, but the quantity obtained was exceedingly small. The Rhine has, for centuries, produced small quantities of gold, and, according to the report of Réaumur, presented to the Academy of Sciences in 1718, its sands have been chiefly worked between Strasburg and Philipsburg. Near Strasburg it was formerly the perquisite of the magistrates of that city, who farmed out the right of gold washing, but in the year above referred to they only received some four or five ounces as their proportion of the annual produce. In 1846, M. Daubrée, a French engineer, made a report to the Academy of Sciences, in which he states that the gravel most usually worked is that deposited below a sandbank or gravel island, which has become eroded by the river, and that gold is only found in any degree concentrated in the coarser gravels, which have been freed from the finer sands by the action of currents.* The gold occurs in the form of small scales or dust, and is constantly accompanied by titaniferous iron, the amount of which is proportionate to the richness of the sand for the precious metal. The workable beds are invariably thin, seldom exceeding from four to six inches in thickness, and the particles of gold remarkably small, since the number required to weigh one milligramme varies from seventeen to twenty-two, and one cubic metre of gravel contains from 4,500 to 36,000 of these scales. Besides the auriferous deposits in the bed of the stream, M. Daubrée states that the ancient detritus on its banks, extending from three to four miles in width, also affords an appreciable amount of gold, but that the fine silt, free from gravel, which is daily accumulating, is totally barren.

The sands of the Rhine are still washed on a small scale, but there is reason to believe that the production was formerly much more considerable than at present. The yield of the year 1846 is estimated by M. Daubrée at 1,800*l.* and the washers usually made from one and a half to two francs per diem, although they occasionally gained from ten to fifteen francs. The same authority estimates the average yield of the sands of the Rhine, Siberia, and Chili to be in the proportion of 1 : 20 : 74; or, if the sand of the Rhine, separated from pebbles of two-thirds of an inch in diameter, be taken as the standard of comparison, the ratio becomes 1 : 10 : 37. In Siberia, sands containing 0·000001 of gold were then not considered worth working; but even this yield is seven and a half times greater than that of the ordinary sands of the Rhine. M. Daubrée, after a careful examination of the subject, came to the conclusion that by the aid of proper appliances these sands might probably be treated advantageously, and goes on to say, that, "by the processes now employed, each washer gains from one and half to two francs a day, and exceptionally even ten or fifteen francs; some of the operations would, however, appear to be susceptible of improvement, since the washing is now entirely effected by manual labour, although there is to be obtained, at the distance of a few steps only, the motive power of the river itself, which, if applied to a dredging machine, could be made to remove the superficial stratum of rich gravel, and deposit it on the head of a washing table."

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* Comptes Rendus, xxii. p. 639, 1846.

Although, however, the application of machinery might be made to materially lessen the expense of working the deposits of the Rhine, the yield of gold is so exceedingly small, that it is questionable whether, by any known mode of treatment, such results could be obtained, as would afford satisfactory returns for the capital and labour which would be required. There are several other localities in France which have afforded small quantities of gold, and the River Ariège (*Aurigera*) is stated to have derived its name from the amount of auriferous sands it deposited. The washing of these is said to have afforded, up to the close of the fifteenth century, an annual produce of about a hundred pounds of the precious metal.

Small quantities of gold have also been collected by washing the *débris* produced by the erosion of some of the lower carboniferous strata in the Department of the Gard, and we have ourselves found traces of gold in the quartzose pebbles, forming one of the constituents of a coarse conglomerate, a little above the village of Bessage.

Spain and Portugal.—Gold mines were successively worked in Spain by the Phœnicians, Romans, and Moors; and although the amount at present obtained from that country is exceedingly insignificant, it at one period produced large quantities of the precious metal. Both Strabo and Pliny speak of Spain as being rich in gold, and mention various localities from which it was obtained. Adrien Paillette, who has investigated the subject of ancient mining in the Peninsula, has arrived at the conclusion, that in former times, both Spain and Portugal yielded large quantities of gold, which was not only obtained from washing the sands of the Duero, Tagus, and some other rivers, but also from workings in the solid rock. It would appear, however, from the investigations of Paillette, that however rich the auriferous regions might have originally been, they had become almost totally exhausted previous to their abandonment; since during the whole of his investigations of the old workings, which are very extensive, he only obtained a few slight traces of gold. Among the gold mines of the Peninsula which have been in operation within more modern times, may be mentioned one near Talavera, worked by Donna Isabella:* another at Adissa, near S. Ubes in Portugal, was worked during the present century, and which produced as much as forty-one pounds weight of gold in 1815.† A mine at Domingo Flores in Leon was worked intermittently from 1639 to 1749.

The present production of the precious metal in Spain is exceedingly small, and is chiefly derived from washing the sands of the rivers Sil and Salor, from which the total annual yield may be estimated at about 1,600*l.*

ITALY.—A great number of localities in this country were known to the ancients as producing gold. The whole district of the Noric Alps (Illyria) was considered highly auriferous, and at one period gold was worked so extensively, that the large quantity produced is said to have caused a reduction of one-third in its price throughout Italy.‡ Gold was also found in Dalmatia,§ the River Po,‖ and at Pithecusa, opposite Cuma.¶

The only gold mines at present of any consequence are situated in Savoy and Piedmont. The chief amalgamation works are built on the small streams near Macugnaga, at the foot of Monte Rosa, where a considerable amount of gold is found in the valleys of Anzasca, Toppa, and Antrona, and to a less extent in those of Alagna,

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* Jacob's History of the Precious Metals, vol. i. p. 272.

† Ibid. p. 283.

‡ Strabo, lib. iv. cap. 6.

§ Plin. Hist. Nat. xxxiii. 21.

‖ Ibid.

¶ Strabo.

Sesia, and Novara. The chief mines in Anzasca are at Peschiera and Minera di Sotto.

The ore is an auriferous pyrites, containing on an average about 12 dwt. of gold per ton. The whole amount produced in the province of Ossola, to which these works belong, was in 1829 about 250 lbs. troy, with a profit of a little over 3,000*l.*[*] In 1844 the yield was as follows:[†]—

	Kilos.	Value.
Valley of Anzasca	153	£16,092
" Toppa	19	2,032
" Antrona	20	2,150
	Total . .	£20,274

The yield of Alagna, Sesia, and Novara during the same year only amounted to 280*l.* Pliny states that these mines were extensively worked in his time, and that the number of slaves who were allowed to work in them was fixed by the Senate at 5,000, in order to prevent a serious reduction in the price of the precious metal.

Several of the mines in this district have recently been undertaken by English companies; but these have not as yet been sufficiently long in operation, to allow of a satisfactory estimate being made relative to the probable returns of gold, although the results hitherto obtained have been of a very encouraging character.

SWITZERLAND produces no amount of gold of any commercial importance; but the sands of the rivers Reuss and Aar, two of the affluents of the Rhine, are known to contain small quantities of this metal.[‡] The Rhine in the Pays de Gex is slightly auriferous, and the Tessin, or Ticino, in the Canton of the same name, deposits sands sometimes affording small particles of gold.

HOLLAND.—Some operations were carried on in this country in the eighteenth century by a Mr. Beecher, with a view to the extraction of gold from the sea sands, but he did not succeed in his enterprise, and a previous adventurer is said to have been equally unsuccessful.

RUSSIAN POSSESSIONS.—The principal portion of the gold from the

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[*] Whitney's Metallic Wealth, p. 96.

[†] *Mining Journal*, 1845, p. 610.

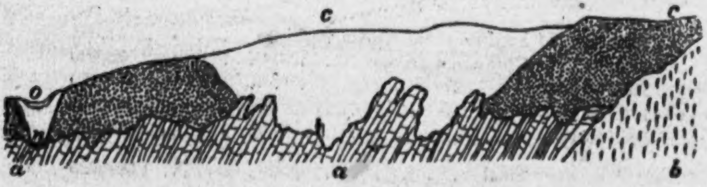
[‡] Dictionnaire des Sciences Naturelles—Or.

Russian Empire is obtained from the western slope of the Ural Mountains, Siberia, and the Caucasus. The government of Archangel also formerly furnished a small quantity of this metal, but the works in that district have been abandoned since the commencement of the present century. In Asiatic Russia, the most productive auriferous districts are comprised within the governments of Perm, Tomsk, Oremburg, Irkoutsk, Yenisseisk, and the district of the Kirghese. The first discovery of gold was made in the year 1743, near Ekatherinburg, and in 1752 the first mining operations were commenced at Berezovsk. These mines, which are sunk in the solid rock, still continue to be productive, although their yield is at present much smaller than it was formerly. In the year 1823 there were no less than sixty-six localities in the Ural from which gold had been obtained by deep mining, but the whole of these undertakings, with the exception of eight, had at that time been abandoned.

The auriferous veins of the Berezovsk mine are numerous, but generally small, and occur in a sort of decomposed granite, which itself forms veins and dykes in talcose, chloritic, and micaceous shales. The most productive veins are of quartz, cutting the granite nearly at right angles, and having an almost perpendicular dip. These veins seldom extend beyond the granite, and appear to become less productive in depth. Towards the close of the last century, the Berezovsk Mine yielded from 600 to 800 lbs. of gold annually, but the production has of late years much decreased, and in 1850 the total yield had been reduced to 100 lbs. troy. During the most prosperous period of their exploitation, the stamped work from these mines afforded from 6 to 11 dwt. of gold per ton.

The gold washings proper of the Ural, which have produced such a large amount of the precious metal, and which, before the discovery of the deposits of California and Australia, had acquired so much celebrity for their richness, were commenced by the Russian Government in 1814, whilst those of Western Siberia were not opened until 1829, and those of Eastern Siberia remained unworked until 1838. The method of occurrence of gold in some of the more important of the Russian alluvial diggings will be understood from a description of the Soimanofsk Mines, north of Miask. Here, great masses of ancient drift or gravel having been removed for the extraction of gold, the eroded edges of highly inclined crystalline limestone have been exposed, which, from being near the centre of the chain, are believed to be of Silurian age. It is from the adjacent eruptive serpentinous masses and slaty rocks, *b*, Fig. 1, that the auriferous shingle *c*, usually more productive near the surface of the abraded rock *a*, has been derived.

FIG. 1.



DIGGINGS AT THE SOIMANOFSK MINES.
(*From "Russia and the Ural Mountains,"* vol. i. p. 487.)

The tops of the highly inclined beds *a* are rounded off, and the interstices between them worn into holes and cavities manifestly by very powerful aqueous action. Here, as at Berezovsk, mammoth remains are found lodged in the lowest part of the excavation, at the spot towards which the small figure of a man is pointing, and at a depth of about 50 feet below the original surface of the overlying coarse gravel *c*, before its removal during the progress of the operations. The feeble influence of the existing stream *n* in excavating even the loose shingle, is seen at *o*, the bed of the rivulet having been lowered by artificial means from its natural level *o*, to that marked *n*, for the convenience of working the deposit. In some spots the gold-bearing alluvium is a heavy clay, whilst in others it is made up of fragments of quartz veins, chloritic and talcose shales, and greenstone, which lie on the side of the hillocks of eruptive rock.

It was from one of these gravelly depressions, south of Miask, that the large lump of solid gold was found, which is now preserved in the Imperial School of Mines at St. Petersburg, and which weighs ninety-seven pounds troy.*

The auriferous detritus of the Ural Mountains is, however, poor in comparison with the deposits which have since been found in California and Australia; for although very large nuggets have occasionally been met with, much of the auriferous ground, which in the

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* Siluria, p. 484.

Russian Empire, where labour is cheap and water-power abundant, can be worked at a profit, would, if in California or Australia, be neglected as being of no practical value.

The following table shows the production of the Russian washings from their commencement to the year 1860:*—

		Lbs. Troy.
1814 to 1820	Produce of the Crown washings	11,085
1820 to 1830	Crown and private	73,200
1830 to 1840	" " "	175,460
1840 to 1850	" " "	553,955
1850 to 1860	" " "	687,025
	Total	1,500,725

The amount of gold obtained from vein mines in the Russian possessions is very small.

The produce of the Russian mines appears to have nearly reached its maximum in 1847, since which there has been but little increase in their returns. Few new localities producing gold have been discovered, and in the Siberian diggings both the yield of the sand and the amount of gold extracted have somewhat diminished. In the Ural, on the other hand, owing to the skill with which the mines are conducted, and the perfection of the machinery employed, the annual produce of gold is increasing, although the yield of the auriferous drift has decreased to 0·00006 per cent. The washing machinery of the Ural is exceedingly efficacious, being capable of treating large quantities of drift at a very small cost; and were this not the case, it would be impossible to work at a profit, since the proportion of gold, in the material operated on, does not exceed from four to six parts in a million.

Germany.—Austrian Empire.—The amount of gold annually produced in Germany is exceedingly small, although in some localities washing and mining operations in pursuit of this metal have been almost uninterruptedly carried on from remote antiquity. Among the rivers of this country which have afforded auriferous sands, may be mentioned the Rhine, the Reuss, the Aar, the Danube, the Elbe, the Moldau, the Oder, and the Weser.

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* Tschewkin Jour. des Mines, quoted in Ann. des Mines (5), iii. p. 805. Archiv für wissenschaftliche Kunde von Russland, 1865, p. 397.

Hungary and Transylvania.—The mines of Hungary have been worked almost without interruption since the eighth century, and are, with but few exceptions, conducted with much skill and economy. In those of Königsberg the gold is disseminated in ores of sulphide of silver, which occur in a partially decomposed feldspathic rock. At Schemnitz, Kremnitz, Neusohl, and Libethen, the treatment of argentiferous and auriferous ores has been carried on for centuries. At Felsöbánya, Kapnick, and Nagybánya, on the western border of Transylvania, there are mines of gold, silver, and copper, which were formerly worked on a very extensive scale. In the mines of Schemnitz, Kremnitz, and Neusohl, the ores afford both silver and gold, together with a sufficient amount of galena to furnish the necessary lead for their metallurgical extraction. The Transylvanian mines afford the rare combination of gold and tellurium, before referred to, but generally yield ores of such low produce as only to admit of being advantageously worked under the united conditions of cheap labour and the application of great mining and metallurgical skill. In Transylvania the gold frequently occurs in veins of considerable magnitude, which have seldom well-defined walls, but abut, without any intervening division, against the enclosing rock. The mine of Kapnik is deserving of notice, from the fact of the gold being frequently associated with orpiment. The produce of gold from the mines of Hungary and Transylvania was, in 1865, 5,395 lbs. troy.

Tyrol and Salzburg.—Small quantities of gold have long been obtained from these districts by the treatment of exceedingly poor ores. At Zell the average yield of the vein-stuff treated was, in 1847, only 4 parts in 1,000,000, equal to about 2¾ dwt. per ton. This is probably the smallest tenure in gold of any auriferous rock mined in any portion of the globe. The annual produce of the mines of Tyrol and Salzburg amounts only to some 65 lbs. troy.

Bohemia.—The mines of Bohemia were worked at a very early period, and it is stated that those of Eula yielded so largely, that, in 734, golden images were manufactured from their produce. There were also gold washings on the Iser, a tributary of the Elbe, within the circles of Bedschow and Turnau, in the Moldau, and at Neuknin, and Bergreichenstein. The produce of the Bohemian mines was from the eleventh to the fifteenth century of some importance to the gold-supply of Europe; but the present total yield does not probably much exceed an annual value of 100*l.*

The different provinces of the Austrian Empire furnished, from the year 1840 to 1847, the following proportional amounts of gold:—

Transylvania	53·30	per cent.
Hungary	45·60	"
Tyrol and Salzburg	0·85	"
Styria, Bohemia, &c.	0·25	"
	100·00	

The total production of the Empire was in 1865 nearly 5,500 lbs. troy.

Central and Southern Asia.—Thibet has often been referred to as being rich in gold, and more particularly the rivers flowing through the western portion of that country are believed to be highly auriferous. Jacob estimates the annual produce of Thibet at 833 lbs. troy.

That portion of the continent of Asia lying south of the Great Himalayan chain is believed to have formerly yielded large quantities of gold; but although a small amount of this metal is still obtained in many localities in India, the country is far less productive than it would appear to have formerly been.

The washings of the Burrampooter are estimated by Jacob to produce annually from 2,500 to 3,300 lbs. troy.[*] These workings, however, afford little or no revenue to the British Government, since those employed in this occupation are but barely enabled to subsist on their earnings, and are consequently too poor to pay any kind of royalty. The sands of some of the streams in the Burmese Empire are irregularly worked by the natives; but, according to Jacob, no reliable data relative to their annual yield can be obtained. Mr. Birkmyre estimates the yearly produce of Ava at about 2,000 lbs.

The Malayan peninsula is known to yield gold, but the amount produced is very small, and the character of the inhabitants such as to prevent the introduction of foreign labour and capital.

China and Japan.—The gold-bearing formations of Siberia extend into China, and were formerly worked to some extent, but, according to Sir R. Murchison, operations have been suspended by the Chinese authorities, in accordance with certain theories relative to the maintenance of the balance of the circulating medium. Japan is also known to produce gold, but, from the difficulty of penetrating into the interior, and the entire absence of statistics, little is known, either respecting the method of occurrence, or the amount annually obtained. Some of the East India Islands also yield gold, and many of them in considerable quantities, but we are entirely without information relative to the amount yearly obtained. Gold washing is extensively carried on in Borneo, particularly on the western coast, where, according to Sir James Brooke, the auriferous beds consist of coarse sand and gravel, varying in thickness from one to four feet, and reposing on a bed of clay ten feet in thickness. The same authority states that there are about 5,000 persons, chiefly Chinese, employed in gold washing on the western coast, and that the value of the gold annually collected exceeds 1,000,000*l*.*

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[*] Jacob's Historical Enquiry into the Production and Consumption of the Precious Metals, ii. p. 330.

Timor, Sumatra, Celebes, and the Philippines are also stated by different authorities to afford gold, but in what amount is difficult to ascertain. Whitney estimates the total production of Southern Asia, including the East India Islands, after making allowance for the small quantity exported from China, at 25,000 lbs.

AFRICA was probably the source of a large proportion of the gold possessed by the ancients, and nearly all modern travellers who have penetrated into the interior of that continent agree in their accounts of its wealth in gold.

The whole of the unmanufactured gold which Africa sends into the market is in the form of dust, evidently obtained from alluvial washings. The mines of Bambouk, south of the Senegal, are the most important, and supply the greater portion of the gold which finds its way to the coast; they are open to all, and are worked by the natives of the surrounding villages. The gold occurs in grains and spangles, in the detritus of ancient streams, and the beds of modern rivers, and is generally mixed with ferruginous sands. In these deposits the negroes frequently dig pits to the depth of from thirty to forty feet, out of which they extract the earth from which they subsequently obtain gold by washing. From Mungo Park's description, it would appear that the gold is principally found in a coarse ferruginous gravel, covered by a stratum of rolled pebbles. Another gold-producing district is that of Kordofan on the Upper Nile, between Darfour and Abyssinia. These mines appear to have been known to the ancients, who believed Ethiopia to abound in gold.

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* "Narrative of Events in Borneo and Celebes," 1848. London.

The auriferous earths are treated by the natives for the metal they contain by washing in wooden bowls, and the gold obtained is usually stored in quills of the vulture, or some other large bird. Russegger, who travelled through Nubia in 1838, arrived at the following conclusions relative to the occurrence of gold in Central Africa.*

The so-called primitive chain, which, in the interior of Africa, stretches from E.N.E. to W.S.W., as well as the alluvium of the rivers flowing from it, is auriferous, but not to the extent popularly believed. The gold of Sennaar and Southern Abyssinia occurs in the form of scales and grains, in quartz enclosed in granite, gneiss, and chloritic slates. In the granite it is found in quartz veins, associated with hematite and iron pyrites, whilst in the stratified rocks it is disseminated, together with various descriptions of iron ore, in immense beds of quartz. In the gold-bearing alluviums, those beds are most productive which are of an ochreous character and intermixed with coarse pebbles; a fine clay, with roots of plants scattered through it, is also frequently auriferous. The more rapid a stream, and the more rocky its bed, the larger is usually the amount of gold found. The gold of Nubia is of a deep yellow colour, and remarkably pure.

Besides the foregoing localities, gold is also collected in small quantities from that portion of the African continent opposite Madagascar, and lying between the twenty-second and twenty-fifth degrees of south latitude.

The total annual amount of gold furnished by Africa is estimated by Dusgate at 3,744, and by Birkmyre at 4,000 lbs.

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* Karsten and Dechen's Archiv. xii. 153, quoted by Whitney.

CHAPTER III.

UNITED STATES OF AMERICA—ATLANTIC OR APPALACHIAN GOLD FIELDS.

DISCOVERY OF GOLD IN THE SOUTHERN STATES—VIRGINIA—NORTH CAROLINA—SOUTH CAROLINA—GEORGIA—TENNESSEE AND ALABAMA—DESCRIPTION OF APPALACHIAN GOLD FIELDS—PRINCIPAL GOLD MINES OF THE SOUTHERN STATES.

THE gold fields of the United States may be divided into two grand geographical sections, viz. those of the Atlantic slope, or the Appalachian gold fields, which have been worked to some extent for the last forty years, and the Californian or Pacific coast gold fields, which, within six years after their discovery, in 1848, had produced more than twelve times the total amount of gold which had, up to that period, been obtained from the regions on the Atlantic shores.

The Appalachian gold fields may be considered to be included within the boundaries of Virginia, North Carolina, South Carolina, Georgia, Tennessee, and Alabama, although some other States have occasionally afforded specimens of the precious metal. The first notice of the discovery of gold in the Southern States which could be found by Professor Whitney, who has carefully examined this subject, occurs in Jefferson's "Notes on Virginia," in which it is stated, that a lump of this metal weighing 17 dwt. had been found near the Rappahannock. Drayton, in his "View of South Carolina," published in 1802, also mentions the finding of a small piece of gold on Paris's Mountain. In 1799 the son of a Mr. Reed is said to have found in Cabarrus County, North Carolina, a piece of gold of the size of a small smoothing-iron, which, after having kept for several years without suspecting its value, he finally sold for about fourteen shillings. Shortly after this, gold was discovered in Montgomery County, and washings were, on a small scale, carried on in these two counties for several years. These operations were entirely confined to the washing of sand and gravel on the banks of various small streams, but several nuggets of considerable size were obtained; and one, found in Cabarrus, weighed 28 lbs. avoirdupois, besides several weighing from 4 to 16 lbs. From this locality it is estimated that over a hundred pounds were collected, previous to 1830, in pieces each over a pound in weight, and in Anson, in 1829, a nugget was found weighing 10 lbs., besides several smaller ones. The first United States gold was coined at the Mint in 1825, and from that time up to 1830, four-fifths of the United States gold coinage was of native gold. From 1804 to 1827, North Carolina furnished the whole of the gold produced in the United States of North America, amounting to $110,000; but in 1829 Virginia contributed $2,500, and in the same year South Carolina yielded $3,500. In 1830 Georgia made its first deposit at the Mint, amounting to no less than $212,000. Previous to 1825 all the gold of North Carolina had been procured from washings, but in that year the auriferous veinstone was discovered *in situ* by Mathias Barringer, who obtained, from an opening about forty feet in length and eighteen in depth, above 625 ounces of gold.

This result turned the attention of miners from the "deposit mines" to "vein mines," and led to the discovery of other leads in Mecklenburg, Guilford, Cabarrus, and Davidson counties. A geological reconnaissance of North Carolina was made by Professor Olmsted about the year 1824, and his observations on the gold fields of that State were published in *Silliman's Journal* in 1825.* At that time the deposit diggings had alone been discovered, and were considered by him to cover an area of 1,000 square miles. Of this region he says that gold, in greater or less abundance, may be found in almost any part of it, and he considers its true bed to be a thin stratum of gravel, which, in low ground, was generally covered by a stratum eight feet in depth of alluvium; when, however, no cause had tended to alter its original depth, it was usually about three feet from the surface. He further remarks that these deposits are not confined to the beds of rivers, but also occur at elevations of 200 feet above the nearest valley. A map of the gold regions was published by Professor Mitchell in 1825, in which nine principal mining localities are laid down, three of which are included by him in the "primary," and six in the "transition" slate.

In Georgia, the first locality known was in Habersham County; but researches were carried on until it was ascertained that the whole of the State lying at the foot of the Blue Ridge was more or less auriferous.

In South Carolina, Brewer's Mine, in Chesterfield, was one of the richest and most extensively worked localities, since in 1830 and 1831 between one and two hundred persons were employed there, and were supposed to average from $1.50 to $3 per day each.

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* Vol. ix. p. 5.

In Georgia the gold excitement was not very lasting; but here, as in the other Southern States producing gold, washing has continued to be carried on by the inhabitants as an occasional occupation.

Tennessee and Alabama have each, during the last thirty years, produced a small annual amount of gold, which has been chiefly obtained from alluvial washings.

Professor Silliman, who visited the gold region of Virginia in 1836, states that the principal mines, then working, were Moss and Busby's, in Goochland County, about fifty miles from Richmond—the Moss vein three-quarters of a mile from Busby's, and the Culpeper Mine on the river Rapidan. This gentleman, after describing the several mining properties of this district, remarks: "In my judgment, nothing could be more inauspicious to the mining interest and the welfare of the country, than a spirit of speculation in these concerns. In an excited state of the public mind, it is rare that facts are correctly reported, or correctly viewed. The speculator, who buys merely that he may sell again, is, too frequently, ignorant of the facts, and reckless of the consequences, in regard to those who may succeed him in his obligations; flattering gains from sales of stock are reported from day to day; the property rapidly changes hands; the public mind, being morbidly excited, is of course blinded, and at no distant period accumulated ruin falls heavily on the last in the train."

Professor Tuomey, in his State Geological Report of South Carolina, in the year 1848, mentions the mines then in course of working; and although a great number of localities were being wrought, the operations must either have been conducted on a very small scale, or at a considerable loss, since the annual produce could not have exceeded about $50,000.

In 1852-53 the discoveries which had recently been made in California produced considerable excitement in the public mind with regard to gold mining generally, and, as a natural consequence, attention was directed to the auriferous districts of the Southern States: many English and American companies were formed for the purpose of working the mines of the Atlantic coast, and for a time great activity prevailed in this region.

But before referring to the individual mines which were in operation at this period, we will very briefly describe the principal characteristics of the Appalachian gold fields.

The Appalachian chain takes its origin in Canada, south-east of the St. Lawrence, and forms a series of mountain ridges extending in a south-westerly direction into Alabama. Its width is variable, being greatest near its centre, and diminishing towards its extremities. Its total length is 1,300 miles, and the most remarkable feature of the system is the number of parallel ridges of which it is composed. Along the south-eastern edge of this series of parallel mountain chains, lies a comparatively narrow undulating range of elevations, known by different names in the various States through which they pass. In Vermont they are known as the Green Mountains, in New York they are called the Highlands, in Pennsylvania the South Mountains, in Virginia the Blue Ridge, and in North Carolina the Smoky Mountains. This belt, which is composed of metamorphosed rocks of the lower palæozoic age, varies from ten to fifteen miles in width, and affords but few organic remains. Immediately to the south of this lies the great auriferous belt, running nearly parallel with the Blue Ridge, and apparently of the same geological age. The central axis of this band has, in Virginia, a direction of about north 32° east; but still further north, follows a line more nearly approaching north and south, and towards the south taking a nearly east and west course. Its width, where most developed, on the borders of North and South Carolina, does not exceed seventy miles. Beyond Maryland, on the north, this auriferous belt is not continuous, a few occasional scales, or small lumps, only, being from time to time met with, until we reach Canada, where a considerable area has been proved to afford a notable amount of gold.

The rocks throughout the whole extent of the auriferous belt present very similar characteristics, and consist of slate of almost every variety, alternating with bands of granite and syenite. The predominating rock is a talcose slate occasionally passing into the chloritic and argillaceous varieties, with a prevailing dip to the east at a very high angle.

VIRGINIA.—The amount of gold produced from this State, although small, had been tolerably constant from 1830 up to the commencement of the late Civil War, when, this part of the country becoming the seat of military operations, regular mining was rendered impossible. The talcose slates which predominate in the gold districts of Virginia have generally a reddish colour, and are closely laminated, their general strike being about 30° east of north. The laminæ for the most part stand almost vertically, and enclose masses of granite, syenite, and protogine. For some depth the auriferous rocks are so much decomposed as to admit of being readily worked by the pick and shovel. The gold is almost invariably found in quartz, which, near the surface, is often cellular, from the decomposition and removal of the iron pyrites with which this metal is so frequently associated. The breadth of the gold-bearing belt in Virginia is about fifteen miles.

Professor Whitney, who describes the gold-mining operations carried on in this State during the great excitement of 1853, enumerates the following as being then among the most important:—

Culpeper Mine, on the river Rapidan, seventeen miles from Fredericksburg: in 1850 this mine was working twelve stamp heads, and two Chilian mills.

Freehold Gold Mining Company, established in England in 1853, after an examination by Mr. W. J. Henwood. Vein twenty feet wide, said to have been traced for a mile and a half.

Liberty Mining Company.—Mine situated in Orange County, seventeen miles from Fredericksburg. The vein worked consists of five parallel bands. Gold contained in quartz, and in the enclosing slates, which are much mixed with decomposing iron pyrites. The auriferous belt sometimes widens out to forty feet. This was an English Company, employing eighteen stamp heads, and six Chilian mills. Average yield of the rock estimated at $8 per ton.

Gardiner Gold Mining Company.—Spottsylvania County. But little work done here (1854); machinery in course of erection, calculated to reduce 100 tons daily. Estimated yield of quartz, $12.50 per ton.

Marshall Mine, said to be paying well; depth 100 feet; also in Spottsylvania County.

Whitehall Mine, same county. According to Mr. Henwood, this mine is in deep blue clay slate, the vein running about south-east and north-west. The veinstone is a hard quartz, sometimes marked by ferruginous stains. In this the gold is scantily disseminated, occasionally associated with ores of tellurium. This district is said by Mr. Henwood to resemble closely that of the Morro de San Vicente of Minas Geraes, Brazil.

Waller Gold Mining Company.—This mine is in Goochland County, and was taken up by an English Company in 1853, under the advice of Professor Ansted, who reported the property to be traversed by the great auriferous belt of Virginia. The veins had been proved to a depth of from five to thirty feet, and the average yield was estimated at 1 oz. per ton.

Garnett and Mosely Mines.—Buckingham County. Worked by an English Company; five or six veins on the property, of which two only were in course of exploration. Forty miners employed, and sixty surface hands; crushing power of seventy-two stamp heads.

London and Virginia Gold and Copper Mining Company.—This Company was formed in London, and incorporated in Virginia in 1853, for the purpose of working the Eldridge Mine in Buckingham County.

Buckingham Gold Company.—Organised in 1853, for the purpose of working a continuation of the vein wrought by the foregoing Company. In describing these mines, Mr. W. J. Henwood remarks:—"In this neighbourhood the lowest visible member of the series is a homogeneous lead-coloured clay slate, rather fissile and somewhat contorted, which is succeeded by white quartzose mica slate closely resembling the itacolumite of Brazil, but of no great thickness. The gold formation follows, and is overlaid by a thin-bedded pale greenish-white talcose slate, which is the upper part of the deposit. The strike of all these beds is about north-east and south-west, and their dip 40° or 50° towards south-east. The auriferous deposit had been wrought to a depth of about twenty-six fathoms in the north-eastern part of the mine; it varies in width from three to twenty feet, and the shallower portions consist, for the most part, of quartz, sometimes vesicular, at others, granular, always very slightly coherent; generally much mixed with earthy brown iron ore: irregularly dispersed throughout the silicious ingredients, are masses of iron and copper pyrites, frequently invested with earthy black copper ore, and vitreous copper. Galena and phosphate of lead occur in like manner, although in much smaller quantities, and in the cellular cavities of the quartz there are numerous grains of fine gold which sometimes present crystalline forms." Tellurium is associated with gold in many of the Virginian mines, and there, as well as in Brazil, the gold is of a high percentage produce.

North Carolina.—After a personal inspection of a portion of the gold districts, Whitney was disposed to believe the chances of successful mining to be greater here than in any of the Atlantic States, provided that companies were established with a view to honestly and systematically testing the value of some of the best-known veins. He, however, evidently doubted the *bonâ fides* of many of the associations then in existence, and appears to have regarded many of them in the light of merely speculative operations.

Gold Hill Mines, in Rowan County, fourteen miles from Salisbury, were the most extensive then worked in the Atlantic States, and are stated to have yielded about $1,500,000 within ten years. The greatest depth to which these mines had been worked was 340 feet.

McCulloch Copper and Gold Mining Company.—The mine formerly worked by this Company was situated in Guildford County, twelve miles from Greensborough. The vein has a known length of about half a mile, and at the time of its purchase by the above Company, in 1853, had been explored over a distance of 1,600 feet. Its average width is about six feet, the enclosing rocks being talcose and micaceous slates. The principal portion of the vein is quartz, and its metalliferous contents, gold, iron, and copper pyrites, and carbonate and oxide of iron. The workings were chiefly above the sixty-feet level, which had been extended 1,200 feet. In 1852, about $31,000 of gold is said to have been taken out at an expense of $60,000. Twenty-five heads of stamps and seven Chilian mills are reported to have been in operation in 1854.

Conrad Hill Mining Company—Davidson County, six miles from Lexington, formerly belonging to Governor Morehead. According to Dr. Genth, there are six known veins on this estate, which are, as usual, composed of quartz, containing a large quantity of disseminated hydrated oxide of iron, together with specular and spathic iron; near the water level, large quantities of copper pyrites occur. There are no statements of the returns obtained from the property.

Vanderburg Mining Company.—Mines situated in Cabarrus County; Company organised in 1853. According to the statements of J. T. Hodge, there are several veins with a course north 50° to 55° east, and which occur in greenstone. One of these had been worked to a depth of 100 feet, and varied in thickness from a few inches to 3½ feet. Large quantities of the rock said to yield $40 per ton.

Phœnix Gold Mining Company.—Mines situated in Cabarrus County. Several estates were owned by this Company, of which the principal is the "Conner tract," on which are three veins, on one of which, the "Sulphur Vein," workings have been carried to a depth of 170 feet. This ledge has been traced for a distance of from 3,000 to 4,000 feet, and has a width of from one to three feet. The "Orchard Vein" has a width ranging from one to six feet, and contains sulphate of baryta and copper pyrites. The average yield of this vein, as far as worked, is stated to have been $20 per ton.

Another vein, occurring in granite, in the same county, in addition to gold, affords specimens of wolfram, scheelite, tungstate of copper, and tungstic acid.

Long and Muse's Mine—Cabarrus County. According to Dr. Genth, there are six or eight veins in this property occurring in dioritic slate, passing into chloritic and talcose slates. Four of these veins he considers to be worth working. The veinstone contains much iron pyrites and galena, which render the amalgamation of the gold by the usual methods somewhat difficult.

Lemmond Mine is situated in Union County, eighteen miles from Concord. Fifteen different veins occur in this property, contained in a very ferruginous talcose and clay slate. Of these, thirteen are quartz veins, and the remaining two belong to the class of fahlband deposits, consisting of beds of slate impregnated with sulphide of iron, frequently decomposed into hydrated oxide, and containing disseminated gold.

Mecklenburg Gold and Copper Company—Mecklenburg County, near Charlotte. This association was formed for the purpose of opening the Rhea and Cathay Mines, which were formerly superficially worked on a cluster of veins which pass through the property, and are said to be highly auriferous. We are without information as to the results obtained from the deeper working of these mines.

SOUTH CAROLINA.—The various gold mines of this State had been almost without exception abandoned, previous to the year 1852, when the excitement caused by the discovery of the Dorn Mine again attracted public attention towards them.

The Dorn Mine is situated at the lower end of the Abbeville district, and was for many years perseveringly, but unsuccessfully, worked by a Mr. Dorn, who, in February, 1852, at length struck a very rich pocket of ore. Up to July, 1853, an amount of gold, equal in value to $300,000, is stated to have been obtained by the aid of a Chilian mill from an opening three hundred feet long, twelve feet deep, and fifteen feet wide. Such rich deposits cannot be expected to be continuous, and it is not surprising that, in 1854, the yield of this property began to decrease. From that period up to the commencement of the late Civil War we are without information relative to the progress of the Dorn Mine. An association was also started in New York, under the name of the Dorn Mining Company, for the purpose of working a property adjoining the original Dorn Mine, but the results obtained do not appear to have been satisfactory.

GEORGIA.—The yield of the gold mines of this State has been for many years gradually falling off; and since the commencement of 1860 but little attention has been bestowed on this branch of industry.*

TENNESSEE AND ALABAMA.—Each of these States has annually produced a little gold, but the amount obtained has been very small.

With regard to the various English Companies formed about the year 1853, for the purpose of working gold mines in the Atlantic States, it is generally known, that they were, without exception, complete failures; and it is believed that the success of the different American Companies, started at the same period, can scarcely be said to have been greater. This result cannot, however, be wondered at when the real object of but too many of these associations is taken into consideration, since it is notorious, that in most instances, the working of the share-market received more attention than that of the mines.

Since the cessation of the recent Civil War, however, considerable attention has been paid to the development of the auriferous quartz veins of the South; and it is believed by many, well qualified to form an opinion on the subject, that some of them would be capable of affording continuous and highly satisfactory returns, if judiciously and honestly worked.

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* For details relative to the gold mines of the Southern States in 1854, consult "Metallic Wealth of the United States."

CHAPTER IV.

UNITED STATES OF AMERICA — GOLD FIELDS OF THE PACIFIC COAST.

DISCOVERY OF GOLD IN CALIFORNIA—SITUATION AND EXTENT OF AURIFEROUS DISTRICTS—SHALLOW AND DEEP PLACERS—QUARTZ VEINS—MARIPOSA—TUOLUMNE—CALAVERAS—AMADOR—EL DORADO—PLACER—NEVADA AND SIERRA COUNTIES, ETC.—STATISTICS OF GOLD PRODUCED IN THE UNITED STATES.

THERE are now within the American possessions west of the Rocky Mountains, three States, and various Territories, embracing a united area of more than a million of square miles, the whole of which may be considered as a gold-mining country; not that the whole of this region is, strictly speaking, auriferous, as there are large tracts lying within it in which the precious metal has never been discovered in any considerable quantity; but still, everywhere scattered over its surface, are found districts, not only rich in gold, but also affording various other important metals. The gold-bearing States west of the Rocky Mountains, are California, Nevada, and Oregon; besides which the territories of Washington, Utah, Montana, Idaho, Arizona, Colorado, &c. annually contribute to the supply of the precious metal obtained from this portion of the globe.

CALIFORNIA.—It had been long stated that gold existed in California, but until this region had been acquired by the United States, comparatively little was known either of the country or of its productions.

The Jesuit Fathers, who virtually governed California during the Mexican occupation, were, doubtless, aware of the auriferous nature of the country, but refrained from encouraging gold washing, as likely to distract the attention of their neophytes, and ultimately defeat the object which brought them into the country. It is a well-authenticated fact, that gold-bearing quartz was worked by a M. Baric, a Frenchman, at a point near the mission of San Fernando, as early as 1843; placer diggings on a small scale, and with moderate results having been engaged in at a much earlier period. An English naval officer is also stated to have brought with him to Europe from that region, more than forty years since, a magnificent specimen of auriferous quartz; and other travellers had, at different periods, reported this portion of the Pacific coast as affording gold.

The first practical discovery of gold in this State was, however, made, either late in February or early in March, 1848: one of the earliest communications published on the subject being a letter from the Rev. C. S. Lyman, who happened to be in California at the time, and who addressed a letter to *Silliman's Journal*, dated San José, March 24, 1848, in which he says: "Gold has been recently found in the Sacramento, near Sutter's Fort. It occurs in small masses in the sands of a new mill-race, and is said to promise well." This discovery was entirely the result of accident. Colonel Sutter, a retired officer of the guard of Charles X., had contracted with a Mr. Marshall for the supply of a certain amount of lumber; and in order to be able to execute this contract, a saw mill was erected on the south fork of the American River, in El Dorado County, at a place now called Coloma. This mill was completed in the spring of 1848, and on setting it to work, the water, rushing rapidly through the tail-race, exposed certain bright metallic particles, which Mr. Marshall recognised as gold. It was at first sought to keep this discovery a secret, but this was soon found impossible; and the news shortly after reaching San Francisco, caused an excitement which quickly emptied it of its inhabitants, then only amounting to a few hundred in number. Colonel Mason, who was at that time Governor of California, was led by the rumours of the discoveries to visit the locality early in the following July, and found that four thousand people were already employed in washing on the American River and its branches, and were extracting gold of the value of from 30,000 to 40,000 dollars daily.

The fame of these extraordinary discoveries was widely spread during the latter months of 1848 and the spring of 1849, and as soon as the dry season had fairly set in, a large influx of immigration commenced. The first to arrive were those who came from the mining districts of Mexico, Chili, Peru, and other States on the Pacific coast, but these were shortly afterwards followed by immigrants from China and the Sandwich Islands, &c. The arrivals from the United States began to be numerous in July and August, and the overwhelming tide of immigration which from that time rapidly flowed into the country, quickly swept away the Mexicans and Chilians, who, to the number of about fifteen thousand, had first occupied the ground.

Shortly after the receipt of Colonel Mason's communication, the United States Government selected Mr. T. B. King to proceed to California as bearer of despatches, and to report on the population, productions, and resources of the country. This gentleman reached San Francisco, June 4th, 1849, and his first official Report is dated March 22, 1850. In the meantime he had made a rapid journey through the then known gold region, and had collected some valuable information; but not being a scientific observer, his opinions relative to the geology of the country were frequently incorrect. The estimates made by him, however, respecting the probable future supply of gold, although considered at the time as highly exaggerated, have in reality been more than justified by the results obtained. Mr. King estimates, that, during the years 1848 and 1849, about 40,000,000 dollars were obtained from the gold washings, and that, at the close of the last-named year, there were nearly 50,000 Americans and 5,000 foreigners engaged in the occupation. This gold was chiefly obtained from the tributaries of the Sacramento, the branches of the San Joaquin not having at that time attracted much attention. The probable yield of gold for 1850 was set down by Mr. King at 50,000,000 dollars, the actual produce, probably, rather exceeding than falling short of that amount.

The great valley of California is drained by two principal rivers and their tributaries, viz. the Sacramento and San Joaquin, the one flowing north and the other south, but uniting about midway, and finally flowing into the ocean through a lateral opening in the mountain range. The extreme length of this great basin is about 450 miles, and its width from 50 to 90.

The Sierra Nevada rises on the eastern limit of this large area, losing its summits in the region of perpetual snows, and having a central axis of granite, through which volcanic matters have occasionally forced their way to the surface. This great granitic mass is flanked on the west by thick bands of slates and shales, alternating with masses of serpentinous and trappean rocks, and extending into the valleys, where they are overlaid by recent deposits of a sedimentary nature.

Towards the west, this great valley is bounded by the "Coast Range," a series of mountains running parallel with the coast, and situated at no great distance from the Pacific shore. The metamorphic slates of the Sierra Nevada form a belt, having a length of above 300 miles, and an average width of 50 or 55 miles, and are the true gold-bearing rocks of this region. The most prolific source from which the gold of California has been derived, is, doubtless, the veins of auriferous quartz enclosed within the slates and other metamorphic rocks, chiefly occurring on the western slope of the Sierra, and which are believed to be not generally older than the Jurassic period. This great original source of the precious metal is, however, in an historical point of view, secondary to the shallow diggings in which the gold was first discovered, as well as to the deeper diggings, often lying beneath volcanic formations, and from which large amounts have been subsequently extracted.

That the placers owe their origin to the degradation and erosion of veins and bands of auriferous rock, and the distribution, by aqueous agency, of the detritus produced, does not appear to admit of controversy. It is also evident that these gold-bearing gravels, although both comparatively modern, are of two very distinct epochs; the latter clearly separated, in point of time, from the former, and its constituents mainly derived from the breaking up and redistribution of the older deposits.

Placer Diggings.—The attention of the first adventurers in California was exclusively directed to the shallow placers in which the gold lay near the surface, and within the reach of miners whose capital consisted of ordinary working tools. Here their labour was often abundantly remunerated, whilst the capital and skill necessary was extremely limited. In proportion, however, as the surface deposits became gradually impoverished and exhausted, appliances were, by degrees, introduced for sluicing and collecting gold, which involved a greater degree of skill, and the employment of a larger amount of capital. Finally, it was discovered that extensive and valuable auriferous deposits were to be found at levels far above the course of the present streams; and in order to treat these effectually, the application of an entirely new system of washing became necessary. To meet these circumstances, the system known as the *hydraulic process* was introduced; but although it has been already in operation for the last ten years, it has, as yet, barely effected any appreciable diminution of the great masses of auriferous shingle, which remain available for treatment by this method of washing.

At length came the era of deep quartz mining, the successful prosecution of which demanded a greater amount of knowledge, and larger capital, than the early resources of California could afford. In carrying out this operation, man undertakes, through his own skill and industry, by breaking up the original matrix, and extracting the precious metal which it encloses, to effect that which, at an earlier period of the world's history, has been done for him by nature on a gigantic scale.

In many localities, and particularly on the ground between the south and middle forks of the Yuba river, these auriferous gravels have frequently, where they have been exposed to denudation, a thickness of 120 feet, and of more than 250 feet where they have been protected by a volcanic capping.*

These vast beds of gravel are composed of rounded masses of greenstone, quartz, and all the metamorphic rocks found above them in the Sierra. They are often locally stratified, but there appears to be no evidence of there being any continuity in the beddings. As a general rule, the lower portions consist of larger boulders than the upper, but this does not exclude the occasional appearance of large rounded masses of rock among the middle and upper members of the series.

When a fresh fracture of the whole thickness of the deposit takes place, such as may be seen in claims which are in active operation, a striking contrast will be often remarked between the colour of the upper and lower portions of this mass of gravel. This is chiefly caused by the oxidation of iron pyrites, enclosed in the upper layers, by the action of the surface water percolating through them, and thus staining the gravels of a red or yellow colour, in undulating bands and seams, which contrast strongly with the blue colour of the unoxidised detritus. On closely examining the blue-coloured portions of these deposits, they will be found to be highly impregnated with finely-divided iron pyrites, forming one of the chief cementing materials, holding together the sand and pebbles in the form of a compact conglomerate, often requiring the use of gunpowder for its removal.

Isolated patches of fine sand, clearly showing water lines, but never regular for any considerable distance, are frequently observed in the upper portions of the beds, and in these are often found large quantities of fossil wood, still retaining its original structure, but flattened by pressure, and blackened to the colour of coal. Sometimes the masses of this ancient drift-wood have so accumulated in the eddies of the former currents, as to present the appearance of an almost continuous bed of lignite. In Calaveras and Tuolumne counties, the volcanic matters, capping the auriferous deposits, occur in the form of basaltic columns, beneath which are found the layers of sand, gravel, and boulders before described. Here the wood contained in the gravel-beds is beautifully silicified and converted into semi-opal, as is also the case at Nevada, Placerville, and elsewhere. It not unfrequently happens, in these localities, that a piece of wood will be observed, which must have been originally converted into lignite at one end, whilst the other remained unaltered; but the whole having subsequently become silicified, it now presents the appearance of a combination of alabaster and black marble, each portion still retaining the structure of the original wood.

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* For information relative to the placers of this district, see Silliman's Report on the Deep Placers of the Yuba River.

The gravel is also occasionally replaced by bands, and lenticular masses, of a tough yellowish or whitish clay, which, by their coherence and plasticity, frequently present a great impediment to the operations of mining. After a few weeks' exposure to the action of atmospheric influences, the agglutinated gravel is often found to disintegrate, and fall to pieces, from the oxidation of the pyritous cement; but in localities in which silica has formed one of the chief constituents of the cementing material, this alteration by weathering is far less rapid in its action. The cementation of gravel is also frequently effected by the agency of calcareous waters.

Gold is, to a greater or less extent, disseminated throughout the whole mass of these great gravel deposits, not, however, with uniformity, but always in greatest abundance near the bottom, and generally in direct contact with the *bed rock*. The upper portions of the gravels are not unfrequently so poor that of themselves they could not possibly be worked with advantage; but as there is no practicable method of washing the lower strata without removing the upper, it is usually the custom to work the whole thickness of the deposit. Large masses of gold are but rarely met with in these ancient gravels, but on reaching the bed rock, the surface of which is often grooved and polished by glacial or aqueous action, and, separating from it, the superincumbent masses of cemented gravel, the gold becomes visible in the form of numerous scales and water-worn grains. These are frequently so inlaid in, and firmly attached to, the bed rock, as to form a sort of mosaic, the whole surface of the bedding requiring to be carefully worked over by the pick, in order to secure the gold attached to, and embedded in, its surface.

In cases where the bed rock consists of a soft material, such as micaceous and chloritic slates, it is in most cases found advantageous to break it up to a depth of from eight to ten inches. The extent of this auriferous alluvium is exceedingly great in California, and in some instances its richness has been remarkable. M. Laur states that "at Mokelumne Hill the last layer was so rich that the laws limited the extent, that each miner could possess, to fifteen square feet. This was exactly the space for a shaft: the working consisted in sinking this shaft down to the unstratified rock; the upper earths, excellent though they were, being thrown away, the workmen confining themselves to a bed of some centimetres in thickness, consisting of a mixture of pyrites and gold, lying immediately on the bed rock. There are shafts in this region which have thus produced 250 pounds weight of gold, extracted from the fifteen square feet constituting their area." *

It must not, however, be supposed that the yield of gold obtained by working equal areas of these deposits, even in the same district, is by any means constant, since certain localities are invariably found to be far more productive than others.

The poverty or richness of a given mass of alluvium, all other conditions being equal, very much depends on the conformation of the bed rock on which it reposes. If the alluvium rests upon rounded convex rocks, the lower beds are generally but slightly auriferous, and the conditions for working are unfavourable. When, on the contrary, these deposits have been formed over depressions in the bed rock, or still better, a gulley or deep hollow is found in the bottom, an abundance of gold may be anticipated. It has also been observed that, other things being equal, the proportionate yield of gold is larger in the southern portion of the Sierra Nevada than in the northern. One of the most remarkable deposits of auriferous gravel is extensively worked in Tuolumne County, under Table Mountain. The summit of this elevation is occupied by a thick bed of basalt of a very dark colour, and great density of texture, which is occasionally distinctly columnar, and appears to have been poured out in one continuous flow. This, in the neighbourhood of Sonora, is from 140 to 150 feet in thickness, and its width, near the entrance of the Buckeye Tunnel, is about 1,700 feet.

Beneath this capping of basaltic lava is a heavy deposit of detrital matter, distinctly stratified in almost horizontal beds, but with a slight inclination from either side towards the centre of the mass. These

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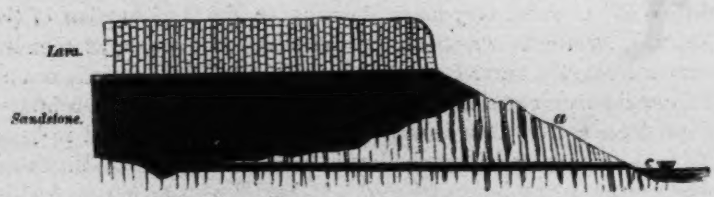
* Du Gisement et de l'Exploitation de l'Or en Californie; par P. Laur, Ingénieur des Mines. Ann. des Mines (VI.), vol. iii. p. 412.

sedimentary beds chiefly consist of a rather fine-grained sandstone, which rapidly disintegrates on exposure to the atmosphere. Interstratified with this sandstone, and more particularly in the proximate vicinity of the bed rock, are clays and fine argillaceous shales, frequently nearly white, and often beautifully laminated. With these are associated beds made up of coarse grains, strongly cohering together, forming the *cement* of the miners; and at bottom is found the *pay gravel*, exactly like that seen in the bed of an ordinary river. The entire thickness of this detrital mass, at its greatest depth, is at least two hundred feet. This thickness, however, diminishes towards the extremities of the deposit, where the edges of the basin formed by the rim rock gradually rise. A good opportunity for measuring the thickness of the various exposed rocks, and for ascertaining the general conformation of the basin, occurs at the Maine Boys' Tunnel near Jeffersonville, the position and relations of which will be understood from the section Fig. 2.

FIG. 2.

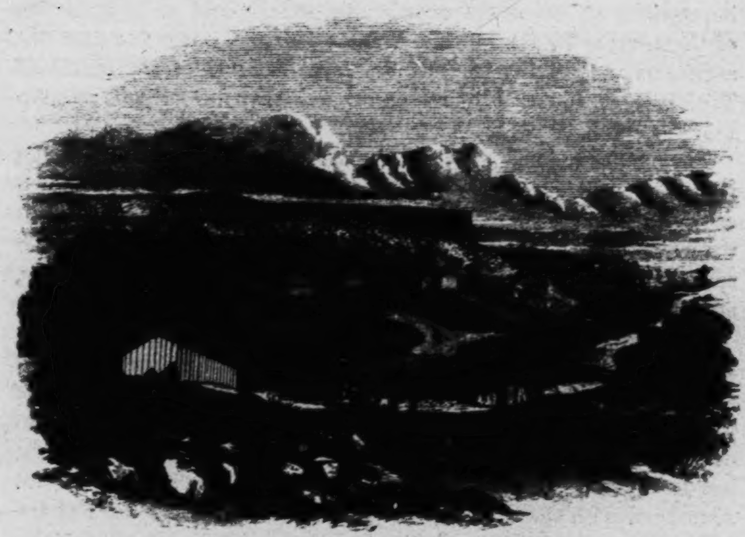


SECTION AT MAINE BOYS' TUNNEL.
(*From "Geological Survey of California," p. 247.*)

The lava is in this place 140 feet in thickness, and the rim rock *a* rises fully 150 feet above the bottom of the channel *b*, which is drained by a tunnel *c*, driven a thousand feet, with only just the amount of declivity necessary to carry off the water. The channel is about a hundred feet wide, and has a band of slate in the middle, standing about three feet above the ordinary level of the bottom. The depth of the auriferous gravel in this channel varies from four to five feet. Some idea will be formed of the great labour and expense of opening up a deposit of this kind, when it is stated that this tunnel was commenced in October, 1855, and that the pay gravel was not reached until March, 1860; the cost of working during that period having been about 9,500*l*.

The general appearance of Table Mountain, with the shelving sides of the auriferous deposit, beneath the cap of basaltic lava, together with the position of the tunnels, will be understood from the annexed woodcut, Fig. 3.

FIG. 3.



TUNNELS UNDER TABLE MOUNTAIN.
(*From "Geological Survey of California,"* p. 249.)

In most instances the tunnels in Table Mountain are driven through the rim rock, with just sufficient fall to admit of the drainage of the workings; but in some cases they are put down on one of the inclined sides of the basin, or through the detrital matter itself, and in such cases the water can only be extracted by pumping. As soon as the channel is reached, galleries are extended, following the streak of auriferous gravel, or pay dirt, lying in the bed of the ancient watercourse. The dirt excavated from these workings is brought in waggons to the mouth of the tunnel, and there tipped into long wooden spouts, from whence it is washed into sluices, by being played on by nozzles attached to a hose, as in the case of hydraulic mining, to be hereafter described.

The stratified deposits, under the lava, frequently contain masses of wood, and even entire trunks of trees, which are almost invariably silicified and converted into semi-opal. In this the vascular structure is often fully preserved, and specimens which have been partially carbonised, before becoming fossilised, are, when polished, exceedingly beautiful. In the more clayey strata of this sedimentary deposit, impressions of leaves are occasionally found; and an examination of these, made by Dr. Newberry, authorises the conclusion that the auriferous strata lying beneath the lava of Table Mountain are of Tertiary age, and that in all probability they belong to the later Pliocene epoch.

Among the animals of the Pliocene of California, or the group which immediately preceded the period of volcanic activity which covered the great auriferous deposit with lava, have been recognised the rhinoceros, an animal allied to the hippopotamus, an extinct species of horse, and an animal resembling the *Megalomeryx* of Leidy, and allied to the camel.

After the close of the volcanic period, a new fauna, belonging to the Post-Pliocene epoch, makes its appearance, and gradually, though slowly, passes into that of the present day. Among the animals of that time were the mastodon and elephant, whose remains are abundantly found in the superficial deposits of the gold region of California, but have never yet been met with in the deposits under the lava. Associated with these were the tapir, buffalo, and two species of the horse—one of which is not to be distinguished from the mustang, or Indian horse, of the present day. It may be here remarked, that the works of man have been so frequently found among the recent deposits of auriferous gravel, and in such close proximity to bones of the mastodon and elephant, as to lead directly to the inference that man existed in this region before the disappearance of these animals from this portion of the earth's surface.*

During the course of the past year, a human skull, in a state of tolerable preservation, is said to have been found embedded in the deep auriferous gravels of California. The circumstances of the discovery of this skull have been partially investigated by Professor Whitney, who is inclined to the opinion that it was really found, as stated, at a depth of 153 feet, beneath five distinct beds of consolidated volcanic tufa, separated from each other by layers of gravel and sand; but as its geological position is lower than that in which the mastodon has yet been discovered in the country, the question of its authenticity becomes one of very great importance, and consequently the results of further investigation will be awaited with considerable impatience.

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* Geological Survey of California, p. 252.

Quartz Veins.—The auriferous quartz veins of California are not equally distributed throughout the entire region of metamorphic slates, but occur chiefly in the vicinity of crystalline and eruptive rocks, forming a sort of fasciculated aggregation, having a width of from ten to fifteen miles, from east to west, and a length, from north to south, corresponding with that of the metamorphic band. These veins, for the most part, follow the general direction of the strata in which they are enclosed. This parallelism is, in many instances, however, rather apparent than real, since not only is the direction of the vein rarely absolutely identical with that of the stratification of the enclosing rock, but quartz veins are invariably associated with branches and offsets, often cutting through the beds of slate at considerable angles. There can, however, be no doubt but that the dip of many of the most important quartz veins in the country is towards the east, and, further, that the veins of the northern counties, Nevada and El Dorado, run nearly north and south, whilst those of Tuolumne and Mariposa have a course almost south-east and north-west. In some instances, the slaty structure is so indistinct, or so entirely wanting, in rocks of the auriferous formation, as to give to them the character of metamorphosed schists or indurated shales.

If we commence the study of the system of quartz veins of the western slope of the Sierra, by beginning towards the southern extremity of the metamorphic rocks, we find, in the county of Mariposa, an enormous central quartz lead, extending from Mount Ophir to beyond Mokelumne Hill, a distance of at least seventy-five miles. This vein, which varies in thickness from six to sixty feet, frequently crops boldly out above the surface of the ground, and can often be seen from the top of an eminence, traversing the country, for miles, like an immense white wall. This lead may be considered as a kind of axis with regard to the other veins of the district, which are chiefly found within a comparatively short distance of it, and have generally an almost similar direction. The great quartz dyke, commencing at Mount Ophir, in Mariposa, can be traced, almost uninterruptedly, to Jackson, in Amador County; but if we now follow on the same line, we reach Folsom and Marysville, where there are no quartz mines in operation; and, in order to again meet with deep mining, we have to turn eastward to the neighbourhood of Placerville.

This northern assemblage of veins is less regular than that of the south, and we do not here find a large representative quartz dyke, readily distinguishable from all the other veins of the district. The veins in this part of the country are more numerous, and smaller in size, than those of the southern mines, and the belt of metamorphic slates is thinner, and more broken up by the protrusion of eruptive rocks. It is consequently more difficult to trace the direction of the axis of this assemblage of veins, than in the case of those of the southern mines; but if we follow the general line of quartz mills near Log Town, Placerville, Volcanoville, Grass Valley, Nevada, and Downieville, these various localities follow each other, from south to north, according to a line bearing about N. 4° west, which may be taken as the mean bearing of the auriferous belt of the northern district.

Having briefly described the general features of the great gold district of California, we will proceed to notice some of the more important operations, carried on in the different mining counties of the Western Slope.

Mariposa County.—One of the most important features of this county is the "Mariposa estate," which contains some of the most celebrated mines of the southern portion of the gold field, and on which large amounts of money have been expended during the last fourteen years. This estate comprehends an area of seventy square miles, and extends from the Merced river south-eastward about sixteen miles, keeping very nearly on the line of the great quartz vein of this district before described. Placer mining has been here carried on since the earliest days of gold digging in the State, and here also, so long ago as 1852, were made the first attempts at quartz mining. The number and magnitude of the outcrops in this district, together with the amount of gold visible in some of them, were, indeed, well calculated to raise extravagant hopes of the boundless wealth to be derived from working them, but these anticipations have, unfortunately, up to the present time, not been realised.

The Pine Tree and Josephine mines are situated near the north-western extremity of the estate, about a mile and a half from the Merced river. The average thickness of the vein is about twelve feet, but it sometimes expands to a width of at least forty feet. This has been worked, chiefly at a height of 1,300 feet above the river, by levels run in on its direction at three different elevations. A portion only of the veinstone is sufficiently productive to pay the expenses of working, but even this is by no means rich. The average yield of the rock from the Pine Tree, was, in 1860, considered to be 12 dwt. per ton, but since that time, the yield has considerably diminished. The Josephine Mine is situated about half a mile south-west of the Pine Tree, probably on the continuation of the same vein, where there is an outcrop of quartz at least twenty feet in thickness, and from which a large quantity of quartz, containing a small amount of gold, has been extracted. That from the Pine Tree and Josephine mines was crushed at the Benton mill, worked by water power, and provided with sixty-four stamp heads, situated on the bank of the Merced river, and yielded, on an average, about 8 dwt. 23 gr. = $8.98 of gold per ton.*

This mill has been in every respect unfortunate, both with regard to its construction and situation; and not long since, the dam, which had been erected, at a great expense, in order to raise the level of the river for the purposes of the establishment, was carried away by a heavy flood. Relative to this establishment, the State Geologist remarks in his recent Report:—"It yet remains to be shown whether, by a system of judicious changes at the mill, combined with careful selection of the quartz, and skill in following the richer shoots, the Josephine and Pine Tree mines can be worked with profit, which at present is not the case, as there has been a considerable falling off in the yield of the quartz extracted since 1860."†

About six miles south-east of the Josephine is another group of veins, occupying a space of about three miles in length, and which, if not continuous, are very closely connected with each other. Of these, the Princeton vein is the most important, having been, a few years since, one of the most productive in California. It was opened to a depth of above 500 feet, and for a length of 1,400 feet. Its direction is nearly that of the enclosing strata, but is subject to many irregularities; its dip, which is invariably towards the east, also changes considerably in amount. The thickness of the vein varies from a few inches to eight feet, and the rock in which it is enclosed consists of a dark-coloured, fine-grained argillaceous shale, which also forms layers in the veinstone itself. The quartz from this locality differs very materially from that of the Pine Tree and Josephine mines, containing more pyrites, galena, and blende, and often showing beautiful specimens of foliated gold. The average yield from this

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[*] Geological Survey of California, p. 228.

[†] It was in this region that Mr. King first discovered Jurassic Fossils.

vein was, in 1860, from $25 to $30 per ton, and it is said that gold, to the value of nearly $2,000,000, has been taken from this mine.

It is, however, understood that the yield of the vein has recently very materially fallen off, and that this is one of the prominent causes of the failure of the Company, recently working the mines on this estate, to render their operations remunerative.

At the north-western extremity of the group of veins, of which the Princeton is a member, is a large outcrop of white quartz, known as the Mount Ophir vein. The workings at this place are not of a very extensive character, but the results obtained have not been sufficiently encouraging to warrant further outlay.[*]

There is another considerable group of veins near the village of Mariposa, on which the first quartz mining in California is said to have been undertaken. The Mariposa vein, which is probably the most important, has been traced for a distance of about two miles, and has afforded some very rich pockets of gold-bearing quartz. The Mariposa estate contains many other outcrops of auriferous quartz on which a large amount of exploratory work has been done; up to the

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[*] The Pine Tree, Josephine, and Mount Ophir veins were worked in 1853-4, by the Nouveau Monde Gold Mining Company, and were reported to be exceedingly rich. No returns of Gold of any considerable amount were, however, forwarded to London, and in 1854 we visited and examined the property. After carefully testing, on a large scale, the quartz from the different locations, we stated their several average yields of gold to be as follows, viz.:

	oz.	dwt.	gr.		
Pine Tree .	0	12	15	per ton of	2,240 lbs.
Josephine .	0	9	17	„	„
Mount Ophir	0	6	0	„	„

Shortly after this, the mines were abandoned by the Anglo-French Company, whose funds had become all but exhausted, and were taken up by the Merced Mining Company, which worked them for a considerable time, and, in the aggregate, obtained a large amount of gold. Subsequently, however, the Fremont claim to the whole district was enforced, and, after considerable litigation, the Merced Mining Company was deprived of its mines.

Some time after Colonel Fremont came into possession of the grant, a Joint Stock Company, on a colossal scale, was organised in New York, for working all the veins in this district, including the Princeton and some others of known value. Owing, however, to unexplained causes, this Company quickly fell into difficulties, and suspended its operations, although there is reason to believe, that, with the present greatly reduced price of labour and materials, this group of veins, if worked on a large scale and judiciously managed, would yield satisfactory returns.

ASSOCIATION, SAN FRANCISCO, CAL.

present time, however, the results obtained have been by no means commensurate with the high expectations, at one time, generally entertained with regard to this property.

Coulterville.—There are several large outcrops of quartz near Coulterville, which are, probably, a continuation of the great quartz vein of which the Pine Tree and Josephine form part. As before stated, this vein appears to run in nearly a straight line from Mount Ophir, on the Mariposa estate, to Jackson, in Amador County, and in the vicinity of the line of these outcrops are found, not only the principal quartz mines, but also the richest surface diggings. There are several localities in the neighbourhood of Coulterville where quartz has been extracted on a small scale, but none of them appear to have afforded results leading to the establishment of permanent works, although some of the outcrops examined are of a very promising nature.

On the banks of the Merced River, near Horseshoe Bend, there is an auriferous quartz vein, possessing considerable interest from the fact of its containing cinnabar in nodules and crystalline plates. The width of this vein, which is enclosed in a highly metamorphosed greenish slate, is about six inches. Its direction is north and south, and its dip towards the west.

McAlpine's.—A few miles north-west of Coulterville a large outcrop of white quartz makes its appearance on the summit of a high hill, known as Peñon Blanco, and forming a conspicuous object for many miles around. On the north side of this great outcrop a quartz mine has been extensively worked on the great quartz lode of the district, and the results obtained are generally understood to be satisfactory. Beneath the productive band there is a thickness of forty feet of totally unproductive white quartz. The quartz above the productive streak differs materially from that beneath it, and contains oxide of iron and laminæ of metamorphic shale. The productive portion of this vein, which varies considerably in width, has been opened for a length of 1,500 feet, and a depth of about 300 feet, at which point it becomes almost vertical, but quite regular. The County of Mariposa, unlike most of the mining regions of California, is without hydraulic diggings, and the shallow placers having become nearly exhausted, the future prosperity of the district must very much depend on the development of its veins of auriferous quartz.

TUOLUMNE COUNTY.—Quartz mining has been carried on at many points in this county, although some of the most productive veins are worked in granite in the vicinity of the metamorphic slates, and not in the slates themselves.

Big Oak Flat.—A large quartz vein passes a little south of this locality, cutting the strata of enclosing slate at an angle; the lines of bedding of the wall rock appearing to run north-west and south-east, whilst the vein itself has a strike of N. 20° W. A considerable amount of mining was done at this point previous to 1863, but in that year the operations were stopped by the burning down of the mill, and we are not aware if they have been since resumed.

Soulsby's.—The Soulsby vein is in granite, is nearly vertical, and runs north and south. The granite in which this vein is enclosed is exceedingly hard, but its walls are well defined; and the lode itself contains a large proportion of the sulphides of iron, lead, copper, and zinc. Near the surface, where the sulphides had become decomposed, the vein, in 1861, gave an average yield of $50 per ton.* In addition to Soulsby's workings, another mine, known as Platt's, is worked on the same vein. Each of these mines has a separate mill; that of the first having twenty stamps, and that of the other, ten. At Lomberdo's Mill, two miles east of Soulsby's, there are several veins occurring in granite, nearly parallel to each other, and running north and south. The only workings on them were, in 1861, on the Louisiana claim, which afforded quartz yielding $20 per ton.

Another group of mills, of considerable local importance, is met with at a distance of about three and a half miles from Soulsby's. The Eureka Mine was here worked in 1861, on a vein in the slates, close to their junction with the granite, bearing N. 25° W. and dipping east at an angle of 40°; the quartz, which was estimated to afford $12 per ton, was crushed in a twenty-stamp mill, driven by water power. The Confidence Mill runs on quartz from a vein in the granite. The width of this vein varies from three to twelve feet, and the average yield is about $15 per ton.

Telegraph Mill.—On the south fork of the Stanislaus, thirteen miles from Sonora, there is a belt of shale, isolated from the main body of the auriferous slate formation, and enclosing a vein of quartz worked at the Telegraph Mill. The thickness of this vein varies from a few inches to twelve feet, and its yield from $15 to $47 per ton; the average yield of one thousand tons of quartz crushed in this mill was $38.80 per ton.

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* This vein for some time yielded $50,000 per month, and although it has not recently kept up to that produce, it has still continued to realise large sums.

Jamestown.—The great main vein, or backbone, of the southern portion of the gold field appears to crop out in Tuolumne county, about a mile below Jamestown, where it forms several eminences, known as Whisky Hill, Poverty Hill, Quartz Hill, &c. &c. The main vein is very large, but irregular; and although it has been repeatedly worked for gold, it has never been found sufficiently rich to pay working expenses. Not far from, and nearly parallel with, the large vein, are several smaller ones, many of which have been profitably worked. One of the most important veins of this district is that of Knox and Company, lying about two hundred feet east of the great main dyke, and running nearly parallel with it. The average thickness of this vein is about twelve inches, and its dip, after the first thirty feet, about 70° towards the east. Beneath the foot wall there is a casing of soft slate, which contains gold, and is often very rich. The average yield of gold from this vein is stated to be $20 per ton. There are several other small mills running in this neighbourhood, but with what results it is difficult to ascertain.

In 1861, a new district was opened up in Sugar-Pine Creek, about twelve miles north-east of Sonora: this when visited by the officers of the State Geological Survey, in 1863, was found to contain some very promising mines. Among them, that which had obtained the largest amount of local celebrity was the Excelsior, on which operations were commenced in 1861. This vein is enclosed in granite, and in 1863 had been explored, by means of an inclined shaft, to the depth of 200 feet. The vein runs N. 40° E., and averages about two feet in thickness. The walls, or casings, of this vein, are rich in gold, and the quartz is said to average from $50 to $75 per ton. It was worked in a ten-stamp mill, on the Tuolumne river; the rock being hauled over the ridge, between the mine and the stream, by steam power.

The quartz-mining interest of this county has, like that of all others in California, been subject to fluctuations; but the operations carried on here do not appear, in the aggregate, to have been attended with that great degree of success attained in some other localities; since, according to Mr. Ashburner's notes, quoted in Geological Report of California, there were in 1859 thirty quartz mills in Tuolumne county, of which about one-half had disappeared previous to 1861; but several new ones have been erected since that time.

The placer diggings of the county of Tuolumne are important, particularly those in, and in the neighbourhood of, Table Mountain, previously described.

Calaveras County.—In the vicinity of Murphy's there is a limestone belt, remarkable for containing quartz veins of sufficient size and richness to admit of their having been extensively worked for gold. At the Blue Wing Mill, there is a vein two feet in width, which runs obliquely across the limestone, the quartz from which has paid as much as $80 per ton. Its run is N. 80° E., and its dip towards the south-east. The great quartz vein of California enters the county of Calaveras at Robinson's Ferry, making itself very conspicuous at Carson Hill, again appearing at Albany Hill, and then at Angel's. It is again seen between that place and San Andreas, and subsequently a little to the north-west of that locality. It has been extensively worked at several points, especially at Carson Hill, and near Angel's.

Angel's Camp was formerly a very active quartz mining centre, there having been at one time thirteen quartz mills at work in the vicinity; but in 1862 only two of them were in operation. The mills at Angel's were all built for the purpose of treating the rock obtained from a lode running parallel with the great quartz vein, and about two hundred feet to the north-east of it. The masses of quartz are parallel with each other, and form somewhat irregular belts enclosed in the talcose slates. A large amount of quartz can be obtained from this place, but its yield in gold is low. The rock crushed at Foster and Company's mill, in 1863, is said to have produced, on an average, $6 per ton. At Carson Hill, a great many exploratory workings have been made; but the gold was found to be very irregularly disseminated. At a place now called the Hope Mine, gold to the value of nearly two million of dollars is reported to have been taken out from a very small space several years since. A band of talcose slates, lying near the back of the great vein, has, in some places, been found exceedingly rich, sometimes paying as much as $80 per ton. Explorations and various attempts at mining have been made on Albany Hill, but hitherto without much apparent success.

West Point.—About sixteen miles east of Mokelumne Hill, near West Point, a great many quartz veins occur in granite. Many of these, which have a course of about N. 50° W. and a westerly dip, have been worked by Mexicans, who grind the quartz in arrastres, and frequently make good wages. These veins are, near the surface, soft, and contain a good deal of oxide of iron, resulting from the decomposition of pyrites; but, as the depth increases, they become harder, and, on reaching the water level, are generally abandoned by the Mexican miners, who transfer their labours to some other outcrop from which quartz can be obtained at a cheaper rate. The Mill vein, which varies from three inches to a foot in width, has afforded quartz, paying as high as $133 per ton. The Myers and Euston lode is very similar to the Mill vein, but is from fourteen inches to two feet in width, and affords quartz yielding satisfactory results. There are also three mills at work on the quartz from the Rathgeber Vein, which is situated three miles east of West Point, and which has afforded some very rich quartz.

The county of Calaveras has been celebrated for the richness of its shallow placers and hydraulic mines. The ordinary river and gulch diggings are now, however, to a very great extent, exhausted, although the surface of the limestone at Murphy's is still being worked, on a rather extensive scale, by means of a long and costly tunnel, which has been brought up at a sufficient depth to command the deepest cavities worn in the rock. Most of the hydraulic diggings, now in operation, are in connexion with the auriferous gravels covered by volcanic deposits, and these are very widely spread over the district.

The deposits beneath the volcanic beds in the vicinity of Mokelumne Hill have been celebrated for their extreme richness, but those in the immediate neighbourhood of the town are now, to a great extent, exhausted. There are others, however, about four miles south, at a place called Chili Gulch, which are still productive, and actively worked. The pay dirt in this place is, as usual, at the bottom of the deposit, and about eight feet in thickness.

Amador County lies between the Cosumnes and Mokelumne rivers, and is considerably smaller than many of the other mining counties. The belt of auriferous rocks, which here occupies a width of about twelve miles, passes directly through its centre, the principal mining towns being Jackson, Sutter, Amador, and Drytown. Ordinary gulch and river mining, although formerly extensively carried on here, is now almost entirely discontinued, whilst quartz mining, on the contrary, has become of great, and rapidly increasing, importance.

Some of the most important quartz mines in the State are situated between Jackson and Drytown, on a belt of quartz which runs nearly north-west from the first-named locality, and which has long been worked with great activity and success. The outcrops, which are by no means those of a continuous vein of quartz, but rather of a series of lenticular masses, occurring in parallel belts, are situated in a line with the great quartz outcrop, commencing in Mariposa County, and which can be traced northward to this place.

The Oneida Mine lies two miles north of Jackson, and is worked on a vein eight feet in width, having the usual strike and dip of the ledges of the district. This mine has been extensively wrought, and affords quartz yielding from $10 to $11 per ton.

Hayward's Mine.—The mines, formerly called the Eureka and Badger, are now generally known as Hayward's, and have been at work for the last twelve years. The vein has a direction of N. 22° W., with a dip of 70° towards the east, and is enclosed in a dark-coloured, rather soft argillaceous slate. Its foot wall is soft and friable, whilst the hanging wall is decidedly harder and more metamorphic. The Eureka and Badger were formerly two distinct concerns, but are now united and worked as one mine. The width of the vein in the Eureka varies from eight to twenty feet, but in the adjoining claim, it suddenly swells out to forty feet. In the two mines, the total length of ground worked is about 500 feet, and to the south of the Badger shaft, which is at the extremity of the claim, nearly all traces of a vein seem to have disappeared, whilst the lode, which is eight feet wide on the north side of the Eureka, pinches out very rapidly in that direction. These mines may therefore be said to be worked rather on a large chimney, or pipe-vein, than on a regular quartz lode. This mine has now reached a depth of more than 1,250 feet, on the underlie of the vein, and still continues to be exceedingly profitable to its proprietors. The quartz is of a greyish-white colour, and is mixed with seams of the enclosing slate, which have a direction parallel to the walls of the vein. The gold is very regularly diffused throughout the veinstone, but is rarely visible to the naked eye. In addition to the free gold, which generally amounts to about $15 per ton, this rock contains nearly three per cent. of auriferous pyrites.

The Eureka Mill has forty stamps, and is worked by water power during the wet season, and by steam when water is not available. The cost of raising the quartz is estimated at $2.50, and the expenses of crushing and amalgamation at $1.32 per ton, thus leaving a large profit on the 90,000 tons annually crushed from the Eureka and Badger mines. The rock from the Badger claim is crushed at the Upper and Lower Badger Mills. The Upper Mill has sixteen stamps, and is run by water; the Lower Mill has but twelve stamps. The stuff from the Badger Mine is very soft, containing a good deal of interstratified talcose slate; it contains about the same amount of gold as the quartz from the Eureka, but, being softer, can be crushed and amalgamated at a cheaper rate. Mr. Ashburner estimates the total cost of these operations at $0.67, which is a lower figure for the treatment of auriferous quartz than can be shown by the books of any other establishment in the State.

The Amador Quartz Mining Company had, in 1862, a twenty-stamp mill driven by steam, and was treating rock from a vein running N. 40° W., with a dip of 57° towards the east. This vein, which has an average thickness of twelve feet, has a hanging wall of soft, black slate, and a foot wall of hard metamorphic rock. At the date above mentioned, it had been worked to a depth of 450 feet on the incline, equal to 377 feet perpendicular; but the workings have been much extended since that time. The yield of the quartz was $10 per ton, and the mill crushed thirty tons daily at an average expense of $1.79 per ton.

The Spring Hill Mills.—There are two mills of this name, situated in Amador

Creek, employed in crushing the quartz of a vein so situated as to allow of the stuff broken, being tipped directly on to the spalling floors behind them.

The average width of the vein is about eighteen inches, although it has in one place suddenly spread out to twenty feet. Its direction is N. 22° W. and its dip towards the east. The average yield of the quartz from this vein is about $10, and the cost of extraction is estimated at $2, whilst the cost of crushing and amalgamation is $1.63 per ton. In some places, this vein is entirely pinched out and replaced by a thin layer of soft black shale, containing finely divided quartz.

Volcano.—Several quartz mines are in operation at this place, and some of them afford rock yielding a good percentage of gold. The Whitman Vein occurs in a band of slate lying between two masses of granite, and is situated about a mile east of Volcano. It can be traced for a distance of above two miles, and has a general direction of N. 22° E. with a dip of 65° towards the west. Its width varies considerably, but may, as an average, be taken at six feet. The Italian's vein occurs in the same band of slate, but its dip is in a contrary direction. The width of this lode is about five feet, and its course N. 40° E. with a dip of 45° towards the east. The Leviathan vein is in the same belt, and has a width ranging from nine to twenty-five feet; its yield is good, although somewhat variable, but seldom declines below $10, or reaches more than $70 per ton. The detrital formation overlying the auriferous slates has a considerable depth in the neighbourhood of Volcano, and is extensively worked for gold. One of the detrital beds in this vicinity is traversed by a distinctly marked quartz vein, passing through the gravel, and evidently formed, since its deposition, by the action of water containing silica in solution. The veinstone is chiefly composed of agate and calcedony, but in some places exhibits ferruginous stains. This is by no means an isolated case, since various other localities have been met with, in which veins of quartz, similar in all their characteristics to those of the auriferous slates, appear to have been formed during the most recent geological period.

El Dorado County lies between Amador on the south, and Placer on the north, comprising that portion of the Sierra extending between the Cosumnes and the middle fork of the American River. The average width of the auriferous belt in this county is considerable, and, at right angles to its direction, must be nearly thirty miles. It would appear from the palæontological evidence, collected by the State Geological Survey, that a portion, at least, of these slates must belong to the Triassic epoch, but this point cannot, as yet, be regarded as being definitely settled with regard to this locality, although it would be impossible not to come to the conclusion that some of the auriferous shales of California belong to that age. A number of fossils have been found in the auriferous slates further north, in Plumas County, which conclusively prove that the slate in that district belongs to the Upper Trias.

This county does not, at present, occupy a very prominent position as a gold-mining region, although it was here that the first discovery of the precious metal was made in 1848. At Logtown, seven miles south of Placerville, there are several veins occurring in the granite, and of which the direction is north-west and south-east, with a dip to the east. There is also a vein at Grizzly Flat, containing gold, associated with large quantities of the sulphides of iron, lead, and zinc. Other quartz veins are known to exist in various parts of the county; but, generally speaking, quartz mining is not extensively carried on. The auriferous detritus covered by volcanic formations does not occupy a large area in El Dorado, but small detached patches are being worked by the hydraulic process.

Placer County lies between Bear River on the north, and the middle fork of the American on the south, and extends from the State line on the east, to the Sacramento River on the west.

Volcanic deposits occupy a large area throughout this county, and consequently its leading interests are connected with hydraulic mining. The known quartz veins are far from numerous, and this branch of industry has not received a large amount of attention.

This county is traversed, from north to south, by an important chain of hydraulic diggings, which are worked on a large scale, the principal mining towns being Iowa Hill, Wisconsin Hill, Yankee Jim's, and Todd's Valley. In many places on this line, the depth of the volcanic and detrital formations is at least five hundred feet, and some of the washings exhibit a vertical section of cement, of more than a hundred feet in thickness. This cement consists of a conglomerate, formed of boulders and coarse gravel, often firmly consolidated by the infiltration of a calcareous or silicious bond, over which lie the usual sedimentary materials, covered by the ordinary capping of hard basaltic lava. At Yankee Jim's, no less than three acres have been stripped of these deposits, and the cemented gravel, which varies in thickness from twenty-five to a hundred feet, has been crushed for its gold. The pebbles and boulders are all thoroughly water-worn, and, to a large extent, consist of quartz; the bed rock is a light-coloured talcose slate, which becomes rapidly decomposed on exposure, and can in many places be readily excavated by the pick and shovel alone.

Much attention has recently been directed to bands of auriferous talcose slates. These deposits are found in the copper-bearing belt, west of the main gold belt of the State, and in the foot-hills of the Sierra. Several of these deposits, now known to contain gold, were first opened as copper mines, the rusty, ochreous outcrops, called among Californian prospecters gossans, or *calico rocks*, from their variegated colours, being supposed to indicate the presence of copper ores in depth. In excavating for copper, these soft ochreous slates were found to contain gold. The Harpending claim, at Whisky Hill, near Lincoln, in this county, is on a vein of this character, and has been worked during the past year on a moderate scale, with a five-stamp mill. The hill rises a hundred feet above the plain, and the metalliferous outcrop is from two to three hundred feet in breadth, and about five hundred in length. Over the whole of this surface the slate outcrops are charged with oxide of iron, and appear in some places almost like blocks of iron ore. On digging down, the rock becomes softer, is more talcose, and, when washed, almost everywhere shows free gold. The material is excavated from an open cut, or quarry, loaded into cars, and conveyed to the mill on a tramway. The softness and abundance of this material allows it to be raised and worked at a very small cost; from thirty-five to forty tons are passed through the mill in twenty-four hours, or at the rate of from seven to eight tons per head. An abundance of water and very coarse grates are used, and the material which passes the batteries is ground in large flat-bottomed pans, known as Moore's grinders. Even with this rude and imperfect way of washing, the material is said to yield from five to six dollars per ton, an amount which it is thought might be much increased by a more effectual system of treatment.

Nevada County comprehends the region lying between Bear River and the Middle Yuba. It is bounded on the east by the State line, and its western boundary is a north and south line, about twenty miles east of, and running parallel with, the course of the Feather River. This county is rich both in quartz mines and hydraulic washings, and mining operations of every description are here carried on, as extensively and successfully as in any portion of California. The width of the auriferous belt in Nevada County is very considerable, but it includes larger areas of granite than any of the counties situated further to the south. Between these masses of granite, the slates are highly metamorphosed, and the sandstones pass into rocks, which can often with difficulty be distinguished from true granite.

The head quarters of quartz mining, is Grass Valley, one of the most prosperous and pleasantly situated towns in the State. The quartz mills in this vicinity are now said to produce about $3,500,000 annually, forming a considerable proportion of the entire produce of the quartz mines in the State. The veins are by no means large, but they more than make up for their want of size by the richness of the quartz which they afford.

Allison Ranch.—The direction of the lode worked on this property is from 5° to 15° west of north, with a dip of from 40° to 45° towards the west. This vein is regular and well defined, its average width being about two and a half feet, with a distinct selvage or flucan on its hanging wall, whilst the foot wall, although distinctly separated from the enclosing rock, is without any clayey division from it. In 1863, the main shaft had been sunk to the depth of 360 feet, on the inclination of the lode; and at the time of our last visit, November 1866, had reached a depth of 500 feet.

The total length of the claim is 1,600 feet, and the ground has been opened, in the direction of the lode, 600 feet north, and 400 feet south, of the central shaft; the average yield of the quartz from the bottom levels being quite equal to that from the upper workings. Mr. Ashburner states, with regard to the produce of the quartz raised from this lode, that 14,858 tons stamped between March 1857 and December 1861, averaged about $50 per ton: some of the rock near the surface, however, which was crushed at the custom mills, in the neighbourhood, is known to have paid as high as $375 per ton. In the early part of 1863, this mine had become, to a considerable extent, impoverished; but in that year a portion of the vein was reached yielding $150 per ton, and since that time, the results obtained continued to be satisfactory up to the commencement of last year, when the yield of the vein again decreased.

The mill has twelve stamps, each weighing 10 cwt. and is driven by steam power, the arrangements for saving gold being similar to those now generally employed in Grass Valley, and which will be described in a future chapter. The Allison Ranch vein contains large quantities of auriferous pyrites, which is carefully collected, and subsequently ground with mercury in a series of cast iron pans. The gross yield of this mine during the last ten years, has been about $2,300,000; the produce for the three years ending December 30th, 1865, having been $1,000,000, but for the past year it has been less than $200,000.

The position of some of the more important quartz mines in the Grass Valley District, together with the direction and dip of the different veins, will be best understood by reference to the Map (Plate I.) kindly prepared by Mr. M. Attwood, Mining Engineer of San Francisco.

The Norambagua Mine has a claim of 3,000 feet, on the course of the vein, which is nine inches in thickness, generally enclosed in syenitic rock, and runs nearly north and south, with a dip of about 15° towards the east. A section of this mine will be found Plate I. fig. 1. Workings have been here extended to a depth of 500 feet, on the inclination of the lode, which is exceedingly flat; and the veinstone, which is a bluish quartz, with interfoliations of a soft blue slate, has been, on an average, found to yield from $65 to $70 per ton. The crushed quartz, on issuing from the mill, is first conducted over riffles, and subsequently passed to the tables of a number of Bradford's separators, by which 1·33 per cent. of auriferous arsenical pyrites is separated and collected. The gold found in this vein is in a very finely divided state, and seldom visible, but the mine has afforded large and regular profits, and has yielded a total produce of more than $1,000,000. The cost of raising the rock is estimated at $30, and the expenses of milling at $5 per ton.

Massachusetts Hill.—The vein here is very flat, and seems to form a kind of basin, being very different, in this respect, to the gold veins usually met with in California. Its greatest thickness is about two and a half feet, and, where best developed, appears to be divided into three distinct bands, with several inches of country rock between each. The proportion of pyrites in this quartz is considerable, amounting to 5 per cent., whilst the average yield of the vein is about $70 per ton. The group of claims on the Massachusetts Hill is stated to have afforded gold to the amount of $3,500,000.

The North Star Mining Company is working on a vein about two miles south-west of Grass Valley, their claim being 3,400 feet in length. This vein, like others in the same district, varies considerably in thickness, but may be taken as having an average width of twenty-five inches. The direction of the lode is nearly east and west, and its dip 27° towards the north. A shaft has been sunk to a depth of 750 feet on the inclination of the vein, and the quartz obtained, by stoping, from that depth, over an extent of 650 feet, is quite equal to any raised from the upper portions of the vein. This mine is worked in a more than usually economical and systematic way, and has reserves of rock amounting to 20,000 tons, yielding on an average, $30 per ton. It has at present in operation, a new mill, consisting of sixteen heavy heads. From 1st January to 1st March of last year (1866), this Company had crushed, in their old mill, 1,400 tons of quartz, which produced gold to the value of nearly $50,000. It is calculated that the reserves in this mine would, of themselves, keep the new stamping mill employed for the next two years.

Eureka Mine.—This mine is situated about one and a half miles north-east of Grass Valley, on a vein more than three feet in thickness, whose direction is nearly east and west, and its dip, south. The ledge has been driven on at the 300 feet level, a distance of 700 feet. This is one of the most valuable mines in California; the gross yield of bullion for the past year amounting to $596,053, and the dividends declared, to $360,000, or an average of $30,000 per month. The company have now on hand about 75 tons of sulphides, worth at least $30,000, and a large amount of wood, timber, and other supplies, valued at $15,000. In addition, $27,000 were expended a short time since for new machinery, and other improvements. It will thus be seen that the earnings of the mine, including actual dividends paid, have for the year 1866 amounted to $432,000. During that period, 12,200 tons of ore were reduced, giving an average yield of more than $48 per ton. A transverse section of this mine will be found Plate I. fig. 2.

The Eureka Mill is probably one of the best in the district, and consists of twenty heads, working in four battery boxes of five heads, each weighing 840 lbs. This machinery usually crushes and amalgamates 43 tons of quartz daily, affording gold to the value of $37 per ton. During the last twelve months, the total cost of raising and crushing the quartz is stated to have amounted to $13.90 per ton.

The Ione Mine is on a vein running north and south, with a dip of 25° to the east. It is enclosed in porphyry, and about two and a half feet in width. The crushing apparatus consists of an eight-head stamping mill, worked by steam, each head weighing 8½ cwt., and crushing 1¾ tons in twenty-four hours. The present perpendicular depth of this mine is 165 feet, the cost per ton of raising the ore $4.50. The total cost of crushing and amalgamation is $1.50, and the average yield of the quartz, hitherto treated, $40 per ton.

Among the more important mines of the neighbourhood of Grass Valley might be mentioned, the Lone Jack, Ophir Hill, and some others, but we have no detailed information respecting these properties. In the neighbourhood of Nevada City there are also some mines and mills of considerable importance.

The Nevada Quartz Mining Company has its mill on the right bank of Deer Creek, about a mile below the town of Nevada, and obtains its quartz from a vein, enclosed in granite, which runs north-west and south-east, and dips at an angle of 45° towards the north-east. The average width of this lode, which contains a considerable amount of sulphides, is about twelve feet. The mill has twelve stamps, driven by water; the average yield of the quartz is said to be $15 per ton, and the net profit for the year 1863, was a little over $67,000.

Sneath, Clay, and Company's Mill is three-fourths of a mile south-west of Nevada, and has twelve stamps run by steam-power. The vein is in close proximity to the mill, and has an average width of two feet. This mill, at the date of its erection, in 1863, was locally regarded as one of the best in the district, and the returns were very satisfactory; but we are without data relative to the recent operations either of this or the Nevada Company.

There are various other mines in this neighbourhood, such as the Star-spangled Banner, and Whigham, affording remunerative returns, but presenting no features of special interest, and relative to which we possess no statistical information. The total value of the gold extracted from quartz mines in the Grass Valley district, since their commencement in 1852, is estimated at above $25,000,000.

This county is not only celebrated for its rich quartz veins, but also for its extensive and productive hydraulic washings, second only to those of Sierra County. It is a question still to be decided, whether the great river bed of the Pliocene Tertiary period of Sierra County, known as the *Blue Lead*, passes through Nevada; but it is known to enter the county at Snow Point: whether, however, it continues its course across it in a northerly direction, to near Red Dog, has not been ascertained. The hydraulic washings near this place are remark-

Fig 2.

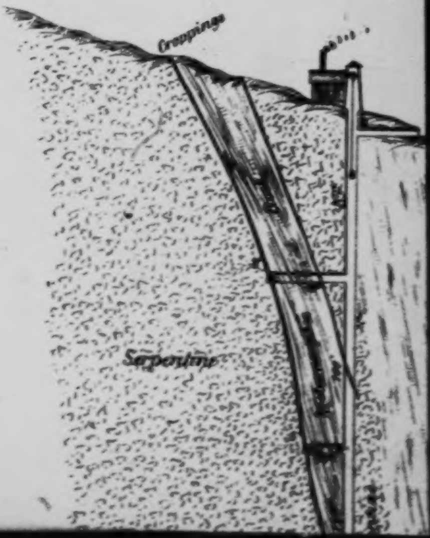
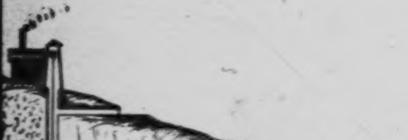
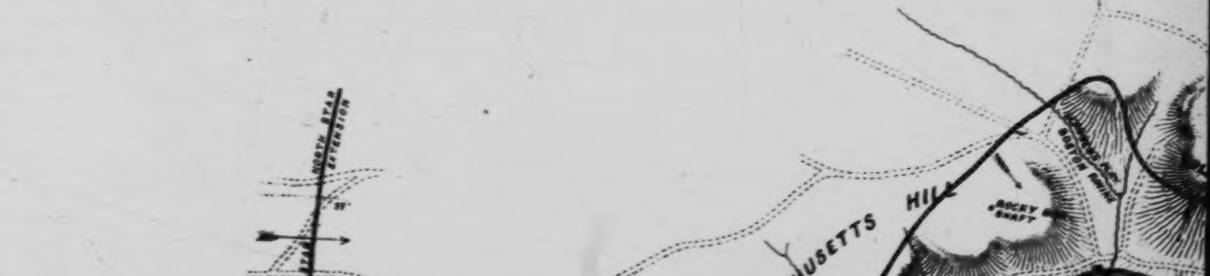


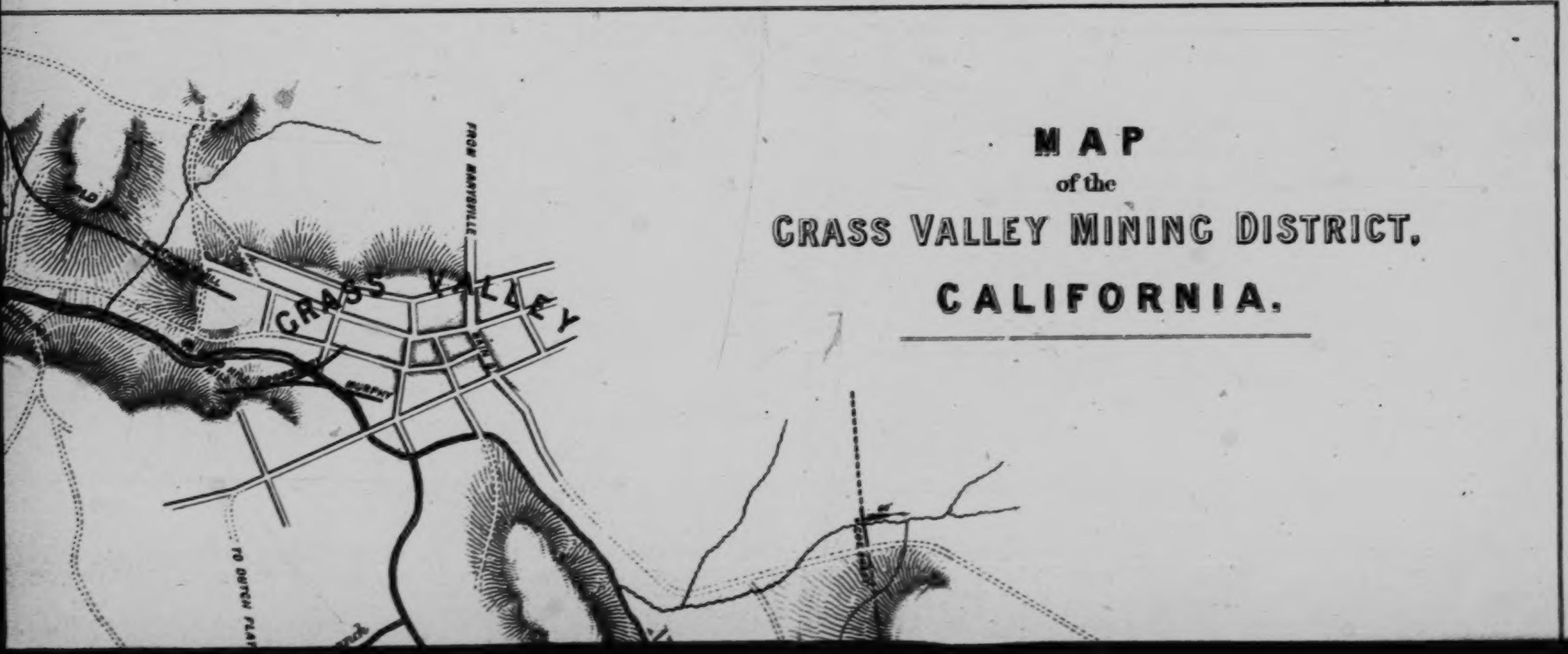
Fig 2.


Metamorphic Rock


NORTH STAR EXTENSION
NORTH STAR
NEW YORK HILL
MASSACHUSETTS HILL
ROCKY SHAFT
BOSTON RAVINE

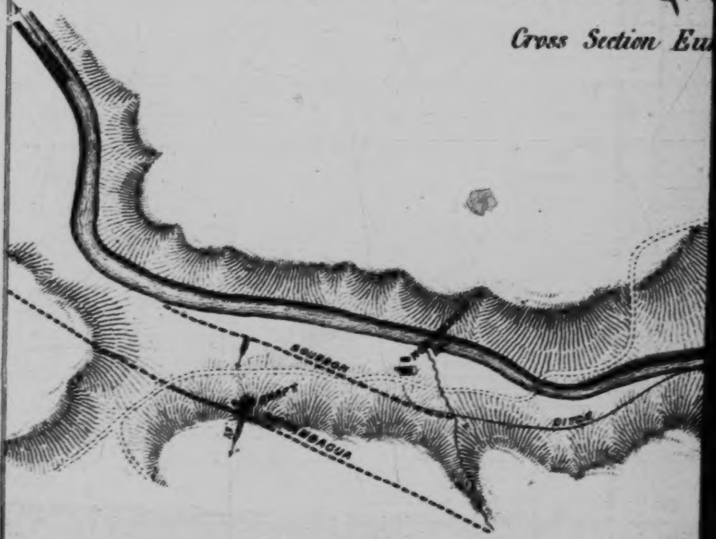
N

PLATE I



MAP
of the
GRASS VALLEY MINING DISTRICT,
CALIFORNIA.

Cross Section Eu



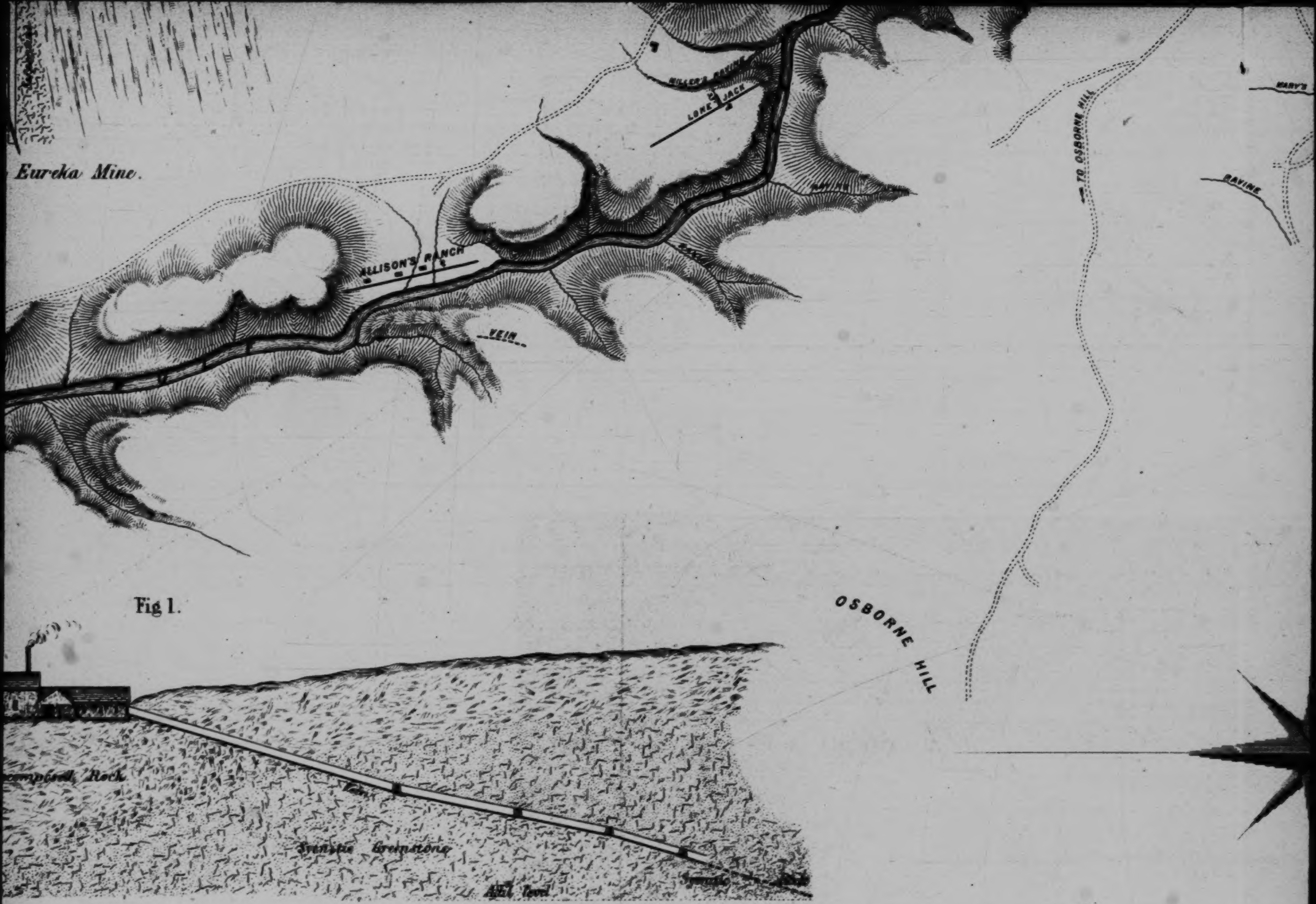
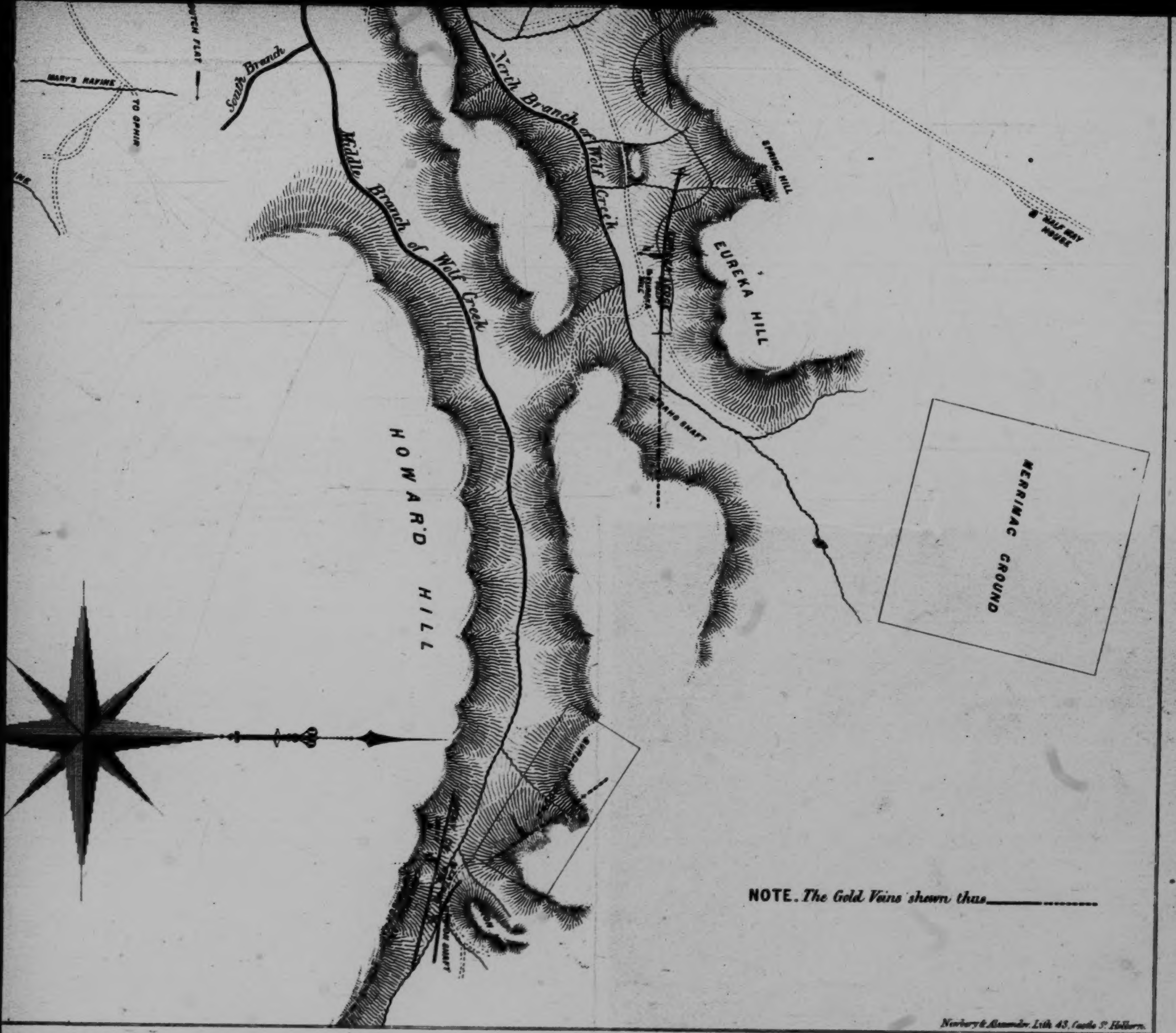


Fig 1.

Cross Section Norambagua Mine

Scale of Map 20 Chains—One Inch.

E. & F.N. Spon. 48, Charing Cross. London.



NOTE. *The Gold Veins shewn thus*

Newbery & Alexander. Lith 43. Castle S^t. Holborn.

able for the large number of trunks of trees buried in the detritus. These are silicified, and bear evidence of having been subjected to the action of powerful currents, before being deposited in the position in which they are now found. The most important hydraulic washings in this county, at the present time, are those extending along its north-western border, parallel to, and in the vicinity of, the bed of the Middle Yuba.

Sierra County is bounded on the south by the Middle Yuba, whilst its northern boundary is formed by the irregularly curved water-shed between the Feather River and North Yuba; east and west it extends from the State line to Yuba County. The western, or principal mining portion of this county covers an area of about twenty-five square miles. Some of the principal quartz mines are situated fourteen miles above Downieville, within 1,500 feet of the summit of the Buttes, and probably about 7,000 feet above the level of the sea.

The Sierra Buttes Mining Company.—The mine and mills belonging to this Company are situated at an elevation of 2,000 feet above the bed of the river, near the outcrop of an immense vein called the Cliff Ledge, which can be traced for a great distance across the ravines by which the country is intersected. The bearing of this ledge is nearly east and west, with a dip of 42° towards the north. It varies from six to thirty feet in width, and is enclosed in a hard metamorphic rock. That portion only of the lode which is found near the foot wall, and which varies from two to seventeen feet in thickness, is passed through the mill, the remainder of the ledge not being considered sufficiently auriferous to pay the expenses of treatment.

This Company has two mills of twelve stamps each, worked by water power, one of them having been built in 1853, and the other in 1856. These mills are capable of working monthly, 900 tons of quartz, of which the yield in 1860-1, averaged $14.82 per ton. The cost of raising the quartz, and delivering it at the mill, is stated to be $5.87, and the expense of crushing and amalgamating $1.35 per ton.

This mine was first taken up in 1851, and worked by arrastres until 1856, when it was purchased by the present owners, who commenced working it in the spring of 1857, with one Chilian mill, and five arrastres, and have obtained the following annual results:—

	Taken from Mine.	Expenses.	Dividends.
1857	$51,000	$15,000	$36,000
1858	55,000	15,000	40,000
1859	88,000	20,000	68,000
1860	120,000	37,000	83,000
1861	198,000	48,000	150,000
1862	166,000	54,000	112,000
1863	156,000	57,000	99,000
1864	90,000	75,000	15,000
1865	196,000	64,000	132,000
	$1,120,000	$385,000	$735,000

The foregoing item of "expenses" includes all improvements, from taking possession of the mine, up to the present time. No calls were ever made, and therefore the whole outlay for machinery has come out of profits realised. In the fall of 1858, the first stamping mill was erected, to which a second was subsequently added. The smallness of the return for 1864, was the result of a deficiency of water, caused by the great drought of that year, and consequently, in order to avoid the recurrence of similar interruptions, an aqueduct was, in 1864, brought in at an expense of $40,000.

In addition to the vein, from which the principal part of the gold produced has been derived, there is on the property, another parallel ledge, called the Aërial, from which some good quartz has been crushed.

The Independence Mine is also on the same veins, and produces quartz having nearly a similar tenure in gold.

A great many other important quartz mines have been opened, and are in full operation in this neighbourhood, but we are without particulars relative to their yield or situation.

The most important feature of this county is the Blue Lead, one of the many ancient river channels found in various localities in the mining regions. The characteristics of this old river bed do not materially differ from those of the auriferous deposits found under Table Mountain, in Tuolumne County, except that in Tuolumne the course of the ancient currents by which the detritus was deposited, was nearly coincident with that of the streams flowing through the district at the present day, whereas in Sierra County the direction of the Blue Lead is nearly at right angles to that of the modern rivers. As a consequence of this difference of direction, its continuity has been frequently interrupted by the erosion caused by the passage through it of the more recent streams. It is also to be observed that the valley of the river under Table Mountain was evidently filled up by one great continuous flow of lava, whilst, in the present instance, the volcanic phenomena are so numerous and complicated, that it is not easy to determine, whether the changes which first turned the course of the ancient river, and subsequently covered, to a great depth, its former bed, were the results of one, or several successive eruptions. Mr. C. S. Capp observes, relative to the Blue Lead: "Hundreds of tunnels have been run in search of it. Where the line it follows has been adhered to, they have always found it, and have been well rewarded for their labour. Millions of dollars have been taken from this lead, and its richness, even in portions longest worked, is yet undiminished. These tunnels have cost from $20,000 to $100,000 each, and interests in the claims they enter, sell readily at from $1,000 to $20,000; in proportion to the amount of ground within them remaining untouched, and the facilities which exist for working it, many of these claims will yet afford from five to ten or more years' profitable labour to their owners, before the lead itself within them is exhausted." *

Yuba County lies west of Sierra County, and extends to the Sacramento River, one-half lying in the mountains, and the other in the plains, the mining district being principally in the former. The chief mining towns are Camptonville, Timbuctoo, Foster's Bar, Texas Bar, and Long's Bar. The Assessor's report for 1860 mentions only two quartz mills as being at work in this county. Hittell states there are twenty-two ditches in Yuba County, having an aggregate length of 942 miles, giving an average of 43 miles each. The most important of these is the Bovyer Ditch, which supplies Timbuctoo, in winter, with 5,000 inches of water; but this amount somewhat diminishes during the summer months.

Butte County lies west of Yuba and Plumas counties, and is drained by the Feather River. Its principal mining towns are Oroville, Bidwell's Bar, Forbestown, Natches, and Whiterock. In 1860 there were twenty-nine quartz mills in this county. It contains sixty-four mining ditches, having an aggregate length of 583 miles. The most important of these is that belonging to the Cape Claim Company, near Oroville, which has an aqueduct, built in 1857, three-quarters of a mile in length, and twenty feet wide, which furnished employment for 250 men during six months, and cost $176,985. The receipts during the first season amounted to $251,426, showing a profit of $74,441. An extension of this flume was undertaken during the succeeding dry season, but the next year's receipts were considerably less than the outlay.

Plumas County.—This is a large, but not very important mining county, and comprises the whole of the region drained by the Upper Feather River and its affluents. About one-third of its surface is more or less auriferous; but, during the winter months, mining operations are much impeded by ice and snow. The principal mining towns are Quincy, Jamison City, Indian Bar, Nelson's Point, and Poorman's Creek. There are five quartz mills in this county, and a considerable amount of gold is obtained by tunnel and hydraulic mining.

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* Quoted in Hittell's "Resources of California," p. 203.

In addition to the gold annually yielded by the great auriferous region which we have thus endeavoured to describe, a certain amount of the precious metal is also procured from various less important localities, in different parts of the State, but which space will not allow us to particularise. We cannot, however, conclude this portion of the subject without noticing the quartz mines of Kernville and Clear Creek, south of Mariposa, whence some patches of the auriferous zone would appear to extend, southward, to those localities. There are several mills at Kernville making satisfactory returns, whilst there is every probability that Clear Creek, at which there are some six or eight already in profitable operation, will eventually become an active and prosperous mining centre.

The gold yield of California reached its culminating point in 1853, and the exportation of treasure, which rose in that year to $57,330,034, gradually fell until 1861, when it was $40,639,080. The silver of Nevada, and the gold of Idaho, then began to flow in, and the amount of the shipments again rose. The following table shows the estimated annual yield of gold, and the annual amount of treasure manifested for exportation, from the commencement of operations in 1848, to 1866, both inclusive:—

Years.	Estimated Amount of Gold produced.	Amount of Treasure exported.
1848	$10,000,000	(no record.)
1849	40,000,000	$4,921,250
1850	50,000,000	27,676,346
1851	55,000,000	42,582,695
1852	60,000,000	46,588,434
1853	65,000,000	57,330,034
1854	60,000,000	51,328,653
1855	55,000,000	45,182,631
1856	55,000,000	48,880,543
1857	55,000,000	48,976,697
Carried on	$505,000,000	$373,467,283
Brot. forwd.	$505,000,000	$373,467,283
1858	50,000,000	47,548,025
1859	50,000,000	47,640,462
1860	45,000,000	42,203,345
1861	40,000,000	40,639,080
1862	34,700,000	42,561,761
1863	30,000,000	46,071,920
1864	26,600,000	55,707,201
1865	28,500,000	44,984,546
1866	26,500,000	44,364,393
Total	$836,300,000	$785,197,016*

About one-third of the gold annually obtained from California is the produce of quartz mining, whilst the remaining two-thirds are procured from the shallow placers and hydraulic mines.

OTHER STATES AND TERRITORIES AFFORDING GOLD.—The gold obtained from the State of Nevada is now chiefly derived from the treatment of the auriferous silver ores of the Great Comstock

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* This includes about $65,000,000, the produce of Nevada, &c.

Vein, in which the gold usually presents about one-third of the total value of the precious metals contained in the rock. The production of gold from this source, since 1861, has been, according to Richthofen, as follows:—

1862 about	$1,500,000
1863 „	4,000,000
1864 „	5,000,000
1865 „	4,750,000
Total produce 1862—1865	$15,250,000

In Eastern Oregon, Washington Territory, Idaho, and Montana, large auriferous districts have been recently discovered, every year adding to their extent, in proportion as these hitherto little known regions become more widely explored. Shallow diggings have been worked for nearly ten years near Fort Colville, on the Upper Columbia river, and results have been obtained which would have been highly satisfactory, had it not been for the difficulty and expense of obtaining supplies, and the frequent troubles with the Indians. The production of gold from these northern countries has been for several years considerable, and is steadily and rapidly increasing; but the whole amount of the precious metal hitherto received from these regions, has been exclusively obtained from river diggings, and other very shallow sources.

It is not, however, probable that, for a very long period at least, these countries will at all approximate in their yield to the annual produce of California, since there are many things calculated to deter any but the most resolute and hardy, from seeking their fortunes in these wilds. The winters are long and severe, and for six months of the year travelling and the transport of materials and provisions are impossible. To this must be added the frequent hostility of Indians, and the notoriously bad social condition of the white inhabitants; life being insecure, and property almost without protection.

A certain amount of gold is annually produced by the territory of Utah, but the quantity is believed to be small. The same may be said of Arizona, although, judging from the reports of miners and others who have visited these regions, it would appear that the limited nature of the returns is rather a result of the small number of miners, chiefly caused by the hostility of the Apaches, than from any deficiency in the richness of the auriferous deposits which have been discovered.

The mining districts on the banks of the Lower Colorado continue to possess attractions for a considerable number of miners, who have been at work on them for several years. Hitherto, however, they have produced but a small amount of bullion, but promise to increase in importance, and will, it is generally believed, ultimately supply the market with large quantities of the precious metals.

The following tables contain valuable data, obtained from official sources, relative to the production of gold in the United States:—

STATEMENT OF GOLD OF DOMESTIC PRODUCTION, DEPOSITED AT THE MINT OF THE UNITED STATES AND BRANCHES, TO THE CLOSE OF THE YEAR ENDING JUNE 30, 1866.

1.—MINT OF THE UNITED STATES, PHILADELPHIA.

PERIOD.	PARTED FROM SILVER.	VIRGINIA.	NORTH CAROLINA.	SOUTH CAROLINA.	GEORGIA.	TENNESSEE.	ALABAMA.	NEW MEXICO.	CALIFORNIA.	NEBRASKA.
1804 to 1827	...	...	$110,000·00	...	...	...	...	...	...	...
1828 to 1837	...	$427,000·00	2,519,500·00	$327,500·00	$1,763,900·00	$12,400·00	...	...	...	...
1838 to 1847	...	518,294·00	1,303,636·00	152,366·00	566,316·00	16,499·00	$45,493·00	...	...	...
1848 to 1857	...	534,491·50	467,237·00	55,626·00	44 577·50	6,669·00	9,451·00	$48,397·00	$226,839,521·62	...
1858	...	18,377·00	15,175·00	300·00	18,365·00	...	...	...	1,372,506·07	...
1859	...	15,720·00	9,305·00	4,675·00	20,190·00	240·00	...	275·00	959,191·79	...
1860	...	17,402·62	8,450·11	...	7,556·41	595·88	92·76	...	663,389·02	...
1861	...	7,209·29	7,523·80	...	15,049·41	...	...	...	426,807·81	...
1862	$68,864·66	...	81·38	...	135·40	...	...	...	244,259·81	...
1863	3,468·69	69·00	1,178·84	...	246·66	...	...	514·53	109,778·58	...
1864	7,896·79	...	6,093·85	...	...	...	...	...	91,663·75	...
1865	7,519·76	910·77	16,298·25	...	10,450·12	...	...	...	64,806·07	...
1866	5,916·98	8,704·64	111,401·39	694·54	37,273·11	...	...	3,155·05	107,024·46	$3,645·08
Total	$93,666·88	$1,548,169·82	$4,575,875·62	$541,161·54	$2,484,059·61	$36,403·88	$55,036·76	$52,341·58	$230,878,450·98	$3,645·08

PERIOD.	MONTANA.	OREGON.	COLORADO.	ARIZONA.	WASHINGTON TERRITORY.	IDAHO TERRITORY.	DACOTAH TERRITORY.	NEVADA.	OTHER SOURCES.	TOTAL.
1804 to 1827	...	...	...	...	...	...	...	...	...	$ 110,000·00
1828 to 1837	...	...	...	...	...	...	...	...	$13,200·00	5,063,500·00
1838 to 1847	...	...	...	...	...	...	...	...	21,037·00	2,623,641·00
1848 to 1857	...	$54,285·00	...	...	...	...	...	...	7,218·00	228,067,473·62
1858	...	3,600·00	...	...	...	...	...	...	...	1,428,323·07
1859	...	2,690·00	$ 145·00	...	...	...	...	...	...	1,012,701·79
1860	...	2,780·16	346,604·05	...	...	...	...	...	1,402·01	1,048,180·26
1861	...	...	607,592·08	$3,048·37	...	...	...	...	1,507·96	1,068,822·46
1862	...	...	1,122,333·50	...	$ 215·70	...	...	...	...	1,435,890·45
1863	...	7,910·78	1,896,329·87	3,869·75	18,563·88	$ 1,816·97	$2,198·88	$103·68	...	2,046,050·11
1864	...	14,192·90	935,146·72	114·72	7,347·97	847,782·60	...	944·74	...	1,911,184·04
1865	$ 453,250·71	11,491·05	308,590·55	276·80	...	1,400,863·12	...	576·37	...	2,274,580·57
1866	1,778,836·84	46,521·12	425,145·14	...	...	286,400·11	...	897·88	...	2,815,616·34
Total	$2,232,087·55	$143,381·01	$5,381,886·91	$7,309·64	$26,127·55	$2,536,862·80	$2,198·88	$2,522·67	$44,364·97	$250,905,913·73

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

2.—BRANCH MINT, SAN FRANCISCO.

PERIOD.	PARTED FROM SILVER.	CALIFORNIA.	COLORADO.	NEVADA.	OREGON.	DACOTAH TERRITORY.	WASHINGTON TERRITORY.	IDAHO TERRITORY.	ARIZONA.	MONTANA.	REFINED GOLD.	TOTAL.
1854	...	$10,842,281·23	...	...	...	...	...	...	...	...	...	$10,842,281·23
1855	...	20,860,437·20	...	...	...	...	...	...	...	...	...	20,860,437·20
1856	...	29,209,218·24	...	...	...	...	...	...	...	...	...	29,209,218·24
1857	...	12,526,826·93	...	...	...	...	...	...	...	...	...	12,526,826·93
1858	...	19,104,369·99	...	...	...	...	...	...	...	...	...	19,104,369·99
1859	...	14,098,564·14	...	...	...	...	...	...	...	...	...	14,098,564·14
1860	...	11,319,913·83	...	...	...	...	...	...	...	...	...	11,319,913·83
1861	...	12,206,382·64	...	...	...	...	...	...	...	...	...	12,206,382·64
1862	$822,823·01	14,029,759·95	$680·00	$13,000·00	$888,000·00	...	...	...	...	...	...	15 754,262·96
1863	1,108,466·57	13,045,711·69	59,472·00	11,250·00	3,001,104·00	$5,760·00	$12,672·00	...	...	...	...	17,244,436·26
1864	220,890·18	14,863,657·52	...	...	2,139,305·00	...	...	$1,257,497·50	...	...	...	18,481,350·20
1865	217,935·98	11,089,974·52	...	5,400·00	1,103,076·54	...	22,460·94	3,499,281·14	$20,369·48	$3,000·00	$2,598,601·49	18 560,100·09
1866	374,393·28	10,034,775·03	...	43,497·28	858,433·11	...	...	2,880,203·48	30,430·68	549,733·32	2,665,033·00	17,436,499·18
Total	$2,744,509·02	$198,231,872·91	$60,152·00	$73,147·28	$7,989,918·65	$5,760·00	$35,132·94	$7,636,982·12	$50,800·16	$552,733·32	$5,263,634·49	$217,644,642·59

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

3.—BRANCH MINT OF NEW ORLEANS.

PERIOD.	North Carolina.	South Carolina.	Georgia.	Tennessee.	Alabama.	California.	Colorado.	Other Sources.	TOTAL.
1838 to 1847 . .	$741·00	$14,306·00	$37,364·00	$1,772·00	$61,903·00	...	...	$3,613·00	$119,699·00
1848 to 1857 . .	...	1,911·00	2,317·00	947·00	15,379·00	$21,606,461·54	...	3,677·00	21,630,692·54
1858	...	...	1,560·00	164·12	...	448,439·84	...	...	450,163·96
1859	...	...	...	...	...	93,272·41	...	...	93,272·41
1860	...	...	...	...	661·53	97,135·00	$1,770·39	...	99,566·92
1861 (to Jan. 31st)	...	...	...	...	...	19,932·10	1,666·81	...	21,598·91
Total . . .	$741·00	$16,217·00	$41,241·00	$2,883·12	$77,943·53	$22,265,240·89	$3,437·20	$7,290·00	$22,414,993·74

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

4.—BRANCH MINT, CHARLOTTE, NORTH CAROLINA.

PERIOD.	NORTH CAROLINA.	SOUTH CAROLINA.	CALIFORNIA.	TOTAL.
1838 to 1847	$1,529,777·00	$143,941·00	...	$1,673,718·00
1848 to 1857	2,503,412·68	222,754·17	$87,321·01	2,813,487·86
1858	170,560·33	5,507·16	...	176,067·49
1859	182,489·61	22,762·71	...	205,252·32
1860	134,491·17	...	...	134,491·17
1861 (to March 31st)	...	65,558·30	...	65,558·30
Total	$4,520,730·79	$460,523·34	$87,321·01	$5,068,575·14

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

5.—BRANCH MINT, DAHLONEGA.

PERIOD.	UTAH.	N. CAROLINA.	S. CAROLINA.	GEORGIA.	TENNESSEE.	ALABAMA.	CALIFORNIA.	COLORADO.	OTHER SOURCES.	TOTAL.
1838 to 1847	...	$64,351·00	$95,427·00	$2,978,353·00	$32,175·00	$47,711·00	...	...	...	$3,218,017·00
1848 to 1857	...	28,278·82	174,811·91	1,159,420·98	9,837·42	11,918·92	$1,124,712·82	...	$951·00	2,509,931·87
1858	...	...	32,322·28	57,891·45	107·33	...	5,293·52	...	...	95,614·58
1859	...	2,656·88	4,610·35	57,023·12	...	...	699·19	$82·70	...	65,072·24
1860	...	3,485·70	2,004·36	35,588·92	...	...	1,097·37	2,490·86	...	44,667·21
1861 (to February 28) .	$145·14	812·79	2,066·91	22,182·14	...	...	4,213·79	32,772·28	...	62,193·05
Total	$145·14	$99,585·19	$311,242·81	$4,310,459·61	$42,119·75	$59,629·92	$1,136,016·69	$35,345·84	$951·00	$5,995,495·95

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

6.—ASSAY OFFICE, NEW YORK.

PERIOD.	PARTED FROM SILVER.	VIRGINIA.	NORTH CAROLINA.	SOUTH CAROLINA.	GEORGIA.	ALABAMA.	NEW MEXICO.	CALIFORNIA.	MONTANA.
1854	...	$167·00	$3,916·00	$395·00	$1,242,00	...	...	$9,221,457·00	...
1855	...	2,370·00	3,750·00	7,620·00	13,100·00	$350·00	...	25,026,896·11	...
1856	...	6,928·00	805·07	4,052·29	41,101·28	233·62	...	16,529,008·90	...
1857	...	1,531·00	1,689·00	2,663·00	10,451·00	1,545·00	...	9,899,957·00	...
1858	...	501·00	7,007·00	6,354·00	12,951·00	2,181·00	...	19,660,531·46	...
1859	...	436·00	20,122·00	700·00	14,756·00	593·00	...	11,694,872·25	...
1860	...	4,202·00	9,755·00	...	19,368·00	...	...	16,023,628·36	...
1861	...	8,869·00	2,753·00	670·30	6,900·00	818·00	$6,714·00	19,227,658·14	...
1862	$241,029·00	316·00	2,282·00	2,065·00	1,469·00	...	1,543·00	12,580,647·83	...
1863	34,828·00	...	130·00	...	...	...	5,580·00	846,244·00	...
1864	7,618·00	...	...	...	...	...	...	116,101·06	...
1865	14,003·00	...	...	...	3,422·00	2,269·00	3,924·00	2,177,954·04	$1,217,518·00
1866	79,304·00	1,693·00	29,536·00	...	11,161·00	1,135·00	...	4,456,392·00	3,182,370·00
Total	$376,282·00	$22,013·00	$81,695·07	$24,519·59	$135,921·28	$9,124·62	$17,761·00	$146,961,848·75	$4,349,888·00

PERIOD.	IDAHO.	COLORADO.	UTAH.	ARIZONA.	OREGON.	NEVADA.	VERMONT.	OTHER SOURCES.	TOTAL.
1854	...	...	...	...	...	...	...	...	$9,227,177·00
1855	...	...	...	...	...	...	...	$1,600·00	25,054,086·11
1856	...	...	...	...	...	...	...	...	16,582,129·16
1857	...	...	...	...	...	...	...	...	9,917,836·00
1858	...	...	...	...	$5,581·00	...	...	27,523·00	19,722,629·46
1859	...	$3,944·00	...	...	2,866·00	...	...	405·00	11,738,694·25
1860	...	248,981·00	$4,680·00	$1,190·00	...	...	...	...	6,311,804·36
1861	...	1,449,166·00	73,734·00	16,871·00	3,181·00	...	...	...	20,759,334·14
1862	...	912,403·00	...	391·00	205·00	$40,846·00	...	3,293·00	13,786,439·83
1863	...	937,535·00	...	391·00	7,813·00	...	$298·00	...	1,332,319·60
1864	$201,288·00	715,208·00	...	3,775·00	8,650·00	74·00	...	117,347·00	1,170,061·06
1865	...	938,593·00	...	707·00	9,876·00	949·00	316·00	364,857·00	4,734,388·04
1866	205,844·00	496,805·00	...	...	8,705·00	5,710·00	...	129,100·00	8,557,664·00
Total	$407,182·00	$5,702,635·00	$78,414·00	$23,325·00	$46,877·00	$47,579·00	$614·00	$644,125·00	$148,894,563·01

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

7.—BRANCH MINT, DENVER.

PERIOD.	COLORADO.	MONTANA.	IDAHO.	OREGON.	ARIZONA.	TOTAL.
1864	$486,329·97	...	...	...	...	$486,329·97
1865	375,065·90	$93,613·01	$71,310·49	$1,230·16	$339·48	541,559·04
1866	96,521·38	44,134·13	19,549·89	777·54	...	160,982·94
Total . . .	$957,917·25	$137,747·14	$90,860·38	$2,007·70	$339·48	$1,188,871·95

STATEMENT OF GOLD OF DOMESTIC PRODUCTION.—*Continued.*

8.—SUMMARY EXHIBIT OF THE ENTIRE DEPOSITS OF DOMESTIC GOLD AT THE MINT OF THE UNITED STATES AND BRANCHES, TO JUNE 30TH, 1866.

MINT.	PARTED FROM SILVER.	VIRGINIA.	NORTH CAROLINA.	SOUTH CAROLINA.	GEORGIA.	ALABAMA.	TENNESSEE.	CALIFORNIA.	COLORADO.	UTAH.	NEBRASKA.
Philadelphia	$5,093,666·88	$1,548,169·82	$4,575,875·62	$541,161·54	$2,484,059·61	$55,036·76	$36,408·88	$230,878,450·08	$5,641,866·91	...	$3,645·08
San Francisco	2,744,509·02	...	...	...	...	...	...	193,231,873·91	60,152·00	...	...
New Orleans	...	...	741·00	16,217·00	41,241·00	77,943·53	2,888·12	22,265,240·89	3,457·20	...	...
Charlotte	...	...	4,520,730·79	460,523·34	...	...	...	87,321·01	...	...	...
Dahlonega	...	...	99,565·19	311,242·81	4,310,459·61	59,629·92	42,119·75	1,136,016·69	35,345·84	$145·14	...
Assay Office	376,282·00	22,013·00	81,095·07	24,519·29	135,921·28	9,124·62	...	136,960,348·75	5,702,635·00	78,414·00	...
Denver	...	...	...	...	...	...	...	...	957,917·25	...	...
Total	$8,214,457·90	$1,570,182·82	$9,278,627·67	$1,353,663·98	$6,971,681·50	$201,734·83	$81,406·75	$584,559,251·23	$12,401,374·20	$78,559·14	$3,645·08

MINT.	MONTANA.	ARIZONA.	NEW MEXICO.	OREGON.	NEVADA.	DACOTAH.	IDAHO.	WASHINGTON.	VERMONT.	OTHER SOURCES.	TOTAL.
Philadelphia	$2,232,067·55	$7,309·64	$52,341·58	$143,741·01	$2,522·67	$2,198·88	$2,536,862·80	$26,127·55	...	$44,364·97	$250,905,912·73
San Francisco	552,733·32	50,800·16	...	7,989,918·65	73,147·28	5,760·00	7,736,952·12	35,132·94	...	5,263,624·49	217,644,642·80
New Orleans	...	...	...	...	...	...	...	...	...	7,290·00	22,414,998·74
Charlotte	...	...	...	...	...	...	...	...	...	...	5,068,575·14
Dahlonega	...	...	...	...	...	...	...	...	...	961·00	5,986,406·95
Assay Office	4,349,868·00	23,325·00	17,761·00	46,877·00	47,579,00	...	407,132·00	...	$614·00	644,125·00	148,928,168·01
Denver	137,747·14	339·48	...	2,007·70	...	...	90,660·38	...	...	...	1,188,671·95
Total	$7,272,456·01	$81,774·28	$70,102·58	$8,182,544·36	$123,248·95	$7,958·88	$10,771,837·30	$61,260·49	$614·00	$5,960,365·46	$652,146,656·41

CHAPTER V.

MEXICO, CENTRAL AMERICA, AND SOUTH AMERICA.

MEXICO PRE-EMINENTLY A SILVER-PRODUCING COUNTRY—NICARAGUA—CHONTALES MINING COMPANY—BRAZIL—ST. JOHN D'EL REY COMPANY—IMPERIAL BRAZILIAN—DON PEDRO NORTH D'EL REY—ROSSA GRANDE GOLD MINING COMPANY—EAST D'EL REY—NEW GRANADA—MARIQUITA AND NEW GRANADA MINING COMPANY—PERU AND BOLIVIA.

MEXICO is pre-eminently a silver-producing country, and almost the whole of the gold which it affords is found in combination with that metal. The amount of silver produced is, however, so large, that the associated gold becomes a matter of considerable importance, and will be more particularly referred to when treating of the ores of silver.

The silver of Guanaxuato and Guadalupe y Calvo is rich in gold, whilst that of Tasco, Catorce, and Zacatecas is poor. In 1840 Duport estimated all the gold produced in Mexico, including that separated from silver, at $\frac{1}{138}$th of the production, by weight, of the latter metal, and its value at $\frac{1}{8}$th of that of the silver.

There are some gold veins in Oaxaca, which have been worked for many years, and which Chevalier was of opinion would at some day become of importance.

CENTRAL AMERICA.

But little is definitely known with regard to the auriferous districts of Central America, except that the gold washings of Costa Rica produce a certain amount of the precious metal, which is mostly smuggled out of the country, and consequently no estimate can be made of its amount. The only European Association professing to carry on gold-mining operations in Central America is, we believe, the following:—

NICARAGUA.—The Chontales Gold and Silver Mining Company.—This Company was formed in 1865 to purchase and work gold and silver mines in the Chontales district of Nicaragua, a gold field hitherto unexplored by Europeans, although worked for several years by the inhabitants. The following description of the properties is condensed from the reports of Captain Paul, the mining agent who was sent out for the purpose of inspecting them, and who has since become the local manager.

The San Domingo Mine comprises an extent of 1,000 varas on the course of the lodes, by 2,000 varas wide. There are two large well-defined auriferous and argentiferous veins running parallel with each other through this sett, about 50 varas apart. One lode has been worked, for about 100 yards in length, to the depth of 20 feet, on its entire width, and is said to have yielded from 1 to 10, and as high as even 100 oz. of gold per ton, while the general average is reported to be about 3 oz. per ton. The hard quartz, yielding nearly 2 oz. per ton, is stated to have been thrown aside, owing to inadequate means for grinding. The mine has, hitherto, been badly worked, and on a small scale, yet 230 oz. of gold were raised in the month of January, 1865.

La Trinidad adjoins San Domingo on a parallel lode. It is 800 varas in length, by 100 in width. The workings are in a deep valley, so that a level can be driven, and 30 to 40 fathoms of backs obtained. This mine has hitherto been owned and worked by natives, who have not had the means of erecting proper machinery, and, in consequence, have been obliged to carry the ore to the mill on men's backs. From 60 tons of ore they obtained 112 oz. of gold.

A shaft has been sunk 27 varas, and levels have been driven at 9, 18, and 27 varas, and in each of them the lode is said to yield an average of 2 oz. per ton. The Cabezales sett, on a continuation of the Javali lode, is 1,200 yards in length, by 200 in width. In one place a shaft is sunk 21 feet deep by 8 feet square, on this lode, and has yielded 30 to 40 oz. of gold, extracted by the most primitive processes. Samples taken from the shaft are stated to average about one ounce per ton, and it is said an abundance of quartz can be obtained when the mine has been properly opened. Javali is situated about one and a half miles from San Domingo. An open cutting has been made in the lode, 60 or 70 yards in length, to the depth of from 5 to 20 feet, and 21 feet wide, still leaving a portion standing. The hard quartz, all of which contains sulphide of silver, and from half to three quarters of an ounce of gold per ton, has hitherto been thrown away, unless it contained visible gold, and there are from 1,500 to 2,000 tons of rock now at surface, which, with proper machinery, would yield a good profit. A new shaft is being sunk in the whole ground to get under the old workings, where the lode yields from 3 to 4, and even up to 40 ounces, per ton. At present only four mills are at work, with which they grind about 160 tons per month, and extract from 200 to 300 oz. of gold; employing 40 to 50 men, most of whom are occupied in carrying the ore to the mill on their backs.

Consuelo is about a mile from the San Domingo, and is 800 yards in length on the course of the lode, by 200 yards wide. A shaft has been sunk about 60 yards deep, and levels driven on the lode, at each eight yards in depth. The lode is not uniformly rich throughout its whole width, but the poorest parts will give at least half an ounce of gold per ton, while the richest, for three feet in width, will average 4 oz. per ton; and taking the rich thread by itself, which is from three to six inches wide, it will yield from 10 to 300 ounces per ton.

San Antonio is on a parallel lode immediately to the north of the San Domingo, and adjoins La Trinidad to the west, being a continuation of the same lode. It is 800 yards in length on the course of the vein, by 200 yards wide. In January, 1865, 50 tons of ore yielded 123 oz. of gold; and from the manner in which the ore was ground and amalgamated, it is believed a large quantity was lost. There are about 500 tons of ore on the surface, which will yield about 1 oz. per ton; but, owing to the distance the present workers have to carry it to the mill, it does not remunerate them.

These mines have not, as yet, made either large or regular returns; but, if the foregoing description of the property be correct, it should, with good management, afford large quantities of bullion at a very early date.

SOUTH AMERICA.

The quantities of gold produced by the mines of South America have never been very large, although they at one time poured forth a stream of wealth, in the form of silver bullion, almost unparalleled in the history of the world. Humboldt estimated, in 1800, the whole produce of gold of the South American Continent at 33,524 lbs., of which 9,900 lbs. were furnished by Brazil; whilst, according to the same authority, the yield of silver during the same period amounted to 691,625 lbs.; the respective weights of the two metals being thus nearly in the ratio of 1 : 29. In the year 1850, the yield of gold had further diminished to about three-fourths of what it was at the beginning of the century, and, according to Chevalier, the total yield of gold from Peru and Bolivia was, in weight, up to 1846, as compared with silver, in the ratio of 1 : 170.

BRAZIL.—This country has long been famous for its gold mines, which have been worked from the beginning of the last century, and have, in the aggregate, produced very considerable amounts of the precious metal. The large quantities of gold produced during the eighteenth century were almost exclusively the produce of the alluvial washings of Minas Geraes, but these have, to a great extent, become exhausted, and the gold now yielded by Brazil is almost entirely the result of deep mining in the solid rock, which is, for the most part, carried on by English capitalists. The auriferous deposits of Brazil differ considerably in their character from those of other parts of the world, since the gold is often rather disseminated in metalliferous beds, than enclosed in regular veins. The enclosing rocks are supposed to be of palæozoic age, but they have been so changed by metamorphic action, as to render it impossible to assign them to any precise epoch. The gold-bearing district consists of a series of disconnected elevations, which do not assume the appearance of a regular mountain-chain, and in which the eruptive rocks come to the surface in dome-like masses. The most generally prevailing formations are gneiss, and those varieties of rock, known in the country as *itacolumite*, *jacotinga*, and *itabirite*. These are characteristic of the gold-bearing rocks of Brazil, and are not generally met with in the mines of either California or Australia, although some of the rocks of the Appalachian gold mines very closely resemble them. Itacolumite is a quartzose rock, intimately incorporated with particles of chlorite, and frequently occurring in bands of enormous thickness. When this rock includes specular iron in its composition, it becomes either itabirite or jacotinga, according as it is crystallised or compact in its structure. Throughout these metalliferous beds gold is disseminated, and those veins are generally most productive which contain deposits of quartz and specular iron.

The most important of the Brazilian gold mines is the Morro Velho Mine, belonging to the St. John d'El Rey Company, where the various operations of mining and reduction are conducted on a very extensive scale. Through the kindness of Mr. Hockin, the Managing Director of this Company, we are enabled to give the following particulars relative to the St. John d'El Rey Company's Mines :—

The St. John d'El Rey Mining Company was first formed in 1830, for working, on lease, the mines of St. Joaõ d'El Rey and St. José near the town of the former name, in the southern part of the province of Minas Gernes, in the Empire of Brazil. These mines, having been found wholly unproductive, were abandoned in 1834, and the Company then purchased the Morro Velho mine and estate, situate at Congonhas, near Sabara, a considerable distance (a degree and a half of latitude) north of the locality of their first operations. The Morro Velho Mine had been previously worked by native proprietors for more than a century, chiefly by open cuttings, and with varied results.

At the time of the purchase of the property by the St. John d'El Rey Company, the Morro Velho Mine was stated to be yielding a profit ; but a considerable outlay having been found necessary, in order to extend the operations and increase the resources of the mine, the Company, having had to expend large sums in the purchase of stock and erection of buildings, &c., worked at a loss during the first four years of its possession of the estate. In 1839, the returns, under the management of the late Mr. C. Herring, to whose judgment the Company is indebted for the selection of the property, again exceeded the outlay ; but the original capital having been exhausted by the losses incurred at the St. John d'El Rey mines, and the purchase of the Morro Velho property, it was found necessary to apply the greater portion of the proceeds of the gold extracted, to the extension of plant, and it was not until 1842 that the first dividend was declared.

From that date, with the exception of an interval of eighteen months on one occasion (1857-8), and twelve months on another (1864-5), during which the working of the most productive portion of the lode was interrupted by a breakage of the pumping and other machinery, the Company has regularly paid dividends every six months.

The original subscribed capital of the Company was	£135,000
Out of which there was returned to the proprietors	6,600
Net paid up capital	£128,400

There has been paid in dividends	£756,245
Laid out in machinery and buildings on the property, out of profits	140,000
Stores existing on the property of the value of	39,000
A reserve fund and working capital has been provided, out of profits, to the extent of	72,249
Making total profit	£1,007,494

The total value of the precious metals extracted from the mine has been	£2,902,480
The total amount of mineral raised	1,769,050 tons.
The average yield of the ore	4·333 oitavas per ton.

Or as nearly as possible half an oz. Troy, value about 32*s.* 6*d.*[*]

The ores from which these results have been obtained, as will be seen by the following figures, have been, on the whole, poor; the yield has been tolerably uniform, the variations that have occurred being attributable rather to the proportion of slate ground with the ore, than to any fluctuation in the quality of the lode itself. Improvements have, from time to time, been made in the mode of treatment, and the loss sustained has been, year by year, steadily reduced.

The average yield of gold per ton, in oitavas, on the whole quantity of stone annually brought to the surface since the year 1847, has been as follows:—

Years.	Oitavas.[†]	Years.	Oitavas.
1848	3·77	1857	3·06
1849	3·89	1858	3·25
1850	4·07	1859	4·13
1851	3·89	1860	4·52
1852	4·25	1861	5·44
1853	4·34	1862	5·92
1854	4·17	1863	5·78
1855	3·98	1864	4·82
1856	3·52	1865	5·47

It should be stated, with reference to these figures, that they do not furnish reliable data whereon to form a judgment as to the increasing or diminishing produce of the auriferous formation in depth, inasmuch as, until recently, no account has been kept of the quantity of clay slate or other unproductive stone raised from the mines. During the last six years, excluding 1864, during which a large portion of the killas or clay slate was stamped with the ore, this unproductive stone has been treated separately, and the proportion it bore to the whole quantity of stuff raised has been as follows, viz.:—

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[*] The gold obtained at Morro Velho is usually alloyed with about 20 per cent. of silver.

[†] An oitava is 2 dwt. 7·343 gr. Troy, or 8·67425 oitavas = 1 oz. Troy.

Years.	Per Cent.	Years.	Per Cent.
1860	18	1863	22
1861	24	1864	—
1862	24	1865	32

Whether during previous years, the quantity of this comparatively unproductive stone was in excess, or otherwise, of the foregoing, there is no means of ascertaining. The formation affording the gold is a strong, well-defined lode, though irregular in direction, dip, and dimensions; its inclination or underlie has also been found to vary at different depths, and in different parts of its extent. The veinstone is mostly composed of quartz with iron pyrites, disseminated, more or less regularly, throughout its mass, and the lode is not unfrequently traversed by clay slate and barren white quartz. When pyrites is absent in these rocks, gold is seldom present.*

In some places the vein is cavernous, and less close in its texture than in others; but where drusy cavities are frequent, the yield of gold diminishes: the most productive matrix for gold is a compact mixture of quartz and pyrites, with varying quantities of slate. The great metalliferous deposit called the Cachoeira, Bahu, and Quebra Panella, is one continuous, very irregular vein, varying in width from seven to seventy feet, and at one point reaching 100 feet. The average thickness at the present depth, 176 fathoms perpendicular on the Cachoeira, and 165 fathoms on the Bahu, is 19 feet: the stoping space extends over 807 square fathoms. There is a north branch, separated from the main deposit by the enclosing rock called the Gamba, but its working, having been found unprofitable, has been discontinued. The enclosing rock is a clay slate of tolerably uniform texture. The shafts, so-called, for the whole of the lode has been excavated from the surface, are carried down at an inclination of about 45°, and the mineral is brought to the surface by tram carriages of a peculiar construction, carrying large kibbes, containing a ton each. The mineral brought to the surface is first freed from slate and other unproductive stone on the spalling floors, and the ore, after being broken to a uniform size, is stamped fine. The rejected slate and quartz is removed by tramways to another establishment, half a mile distant, and there employed to assist in the further pulverisation of the refuse sand from the first stamping, which is re-stamped.

The stamping mills, as is also the pumping and other machinery, are moved by water-power. The pulverised ore, issuing from the stamp coffers, through finely perforated copper grates, passes over bullock skins, in the first instance, and lower down the inclined tables, over woollen cloths. The bullock skins are taken up and washed in vats every hour, and the woollen cloths at longer intervals. The concentrated sand resulting from washing the bullock skins, is subsequently amalgamated in barrels.

The subjoined table shows the quantities of rock raised and stamped, the amounts of gold produced, and annual net profits made since 1848.

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* Arsenical, magnetic, and ordinary iron pyrites predominate at different points, and in varying quantities; carbonate of lime, dolomite brown spar, and, very rarely, copper pyrites, are also present in the vein.

	1849.	1850.	1851.	1852.	1853.	1854.	1855.	1856.	1857.
Stone raised, tons	67,336	67,106	79,810	82,642	85,696	86,048	87,297	89,677	86,407
Stone and Ore stamped, tons	69,004	64,318	81,629	31,236	86,866	96,423	86,848	87,424	86,335
Gold produced, lbs. Troy	2,583	2,517	3,057	3,323	3,623	3,464	3,325	2,992	2,589
	£	£	£	£	£	£	£	£	£
Net Profit	18,136	35,880	51,586	55,391	49,273	44,740	34,466	23,233	787

	1858.	1859.	1860.	1861.	1862.	1863.	1864.	1865.
Stone raised, tons	88,901	88,968	91,361	96,612	90,696	84,758	65,435	78,883
Stone and Ore stamped, tons	87,270	82,880	74,523	71,962	67,508	65,697	62,147	59,607
Gold produced, lbs. Troy	2,733	3,294	3,974	5,651	5,182	4,718	2,852	4,153
	£	£	£	£	£	£	£	£
Net Profit	8,545	38,058	60,460	96,769	87,531	63,285	...	80,438
Loss	...	...	...	...	...	...	14,629	...

Since 1860 the slate and other unproductive stone has been rejected, and the ore only stamped. The clay slate thus separated is crushed in another department of the works. In the above table the profit for that year has been calculated on accounts made up to the end of February, whilst the other figures given are the result of operations up to the end of December in each year. This company employs upwards of 2,400 hands, from 120 to 130 of whom are Europeans. The number of stamp heads at work is 135, for reducing the ore in the first instance, and 56 for re-stamping the residual sand, with slate and quartz: arrastres are also used for re-pulverising the residual sand, and are found very efficient for that purpose. The work performed by the stamps is given in the following table:—

1865.

TABLE OF STAMPS DUTY.

STAMPS.	Heads.	Blows per Min.	Days working.	Tons of Ore stamped.	Tons per Day.	Pounds per Head per Day.
Lyon	30	56	357·46	12,100·6	33·85	2,527
Cotesworth	12	59	356·00	4,932·1	13·85	2,585
Susanna	9	60	361·12	2,897·2	8·02	1,996
Herring	24	77	357·29	11,495·0	31·89	2,976
Powles	36	64	351·24	17,562·6	50·00	3,111
Addison	24	67	353·52	10,619·8	30·04	2,803
Total	135	—	—	59,607·3	167·65	—

Imperial Brazilian Mining Association.—This Company was formed in 1825, for the purpose of working the Gongo Soco and other mines in the province of Minas Geraes, and in the course of fifteen years produced nearly a million sterling. The following is a statement of the financial operations of the Imperial Brazilian Mining Association at Gongo Soco, from 1st January, 1826, to 31st December, 1856:*—

RECEIPTS.			
Proceeds of gold dust			£1,467,448
PAYMENTS.			
Salaries and wages	£432,942		
Materials	451,995		
		£884,937	
Provincial duties paid Brazilian Government	£310,777		
Export duties, ditto	22,403		
		333,180	
			1,218,117
Actual profits			£249,331

Gold was discovered at the surface 122 fathoms above the bottom of the mines, but, owing to the slope of the ground, the deepest shaft was only 56 fathoms in depth; and for the last eight or ten fathoms, the vein, which had entirely changed its character, afforded only a few particles of gold.†

Don Pedro North d'El Rey.—This Company was, on the recommendation of Captain Treloar, formed in 1863, for the purpose of purchasing and working the mine of Morro de Santa Anna. This gold mine forms a portion of the celebrated range of mountains called the Sierra de Itacolumi. The mine is about six miles north from Ouro Preto, the capital of the province of Minas Geraes, and two miles, west, from the cathedral city of Marianna. It is distant from the St. John d'El Rey Company's mine, Morro Velho, about forty miles in a southerly and easterly direction. The holding, or partition of the mining concession in this locality, differs from that of similar property generally in Minas Geraes. In 1762, permission was given by the Government to open mines in this mountain, wherever the miners might feel inclined, and on doing so they became the lawful possessors of fifty palmos of ground on each side of their levels and shafts. Owing to this, the mountain is studded with mines, which the owners from time to time went on selling, until, with a few unimportant exceptions, the whole became the property of the present Company. At one period, upwards of 5,000 miners were working on this mountain, and, in order to extract the precious metal, most of them crushed the mineral obtained by hand, and it must therefore have been rich to have repaid them for their trouble.

The mountain of Morro de Santa Anna rises some 2,000 feet above the valley, but behind it there are still higher mountains, and the mine is consequently well provided with water; water is brought in at different levels, and more could be obtained if required. Near the summit the mountain has been, in most places, completely honeycombed; but downwards, the auriferous formations are entire for a considerable elevation above the valley. The Morro de Santa Anna contains three auriferous formations; two of them are of jacotinga, and the third is a rock formation. All three have yielded large quantities of gold, but the rock, with the jacotinga next it, have been most productive. The face of the mountain is covered with *canga*, an iron conglomerate, about four feet thick; this is auriferous, and will probably pay for stamping.

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* Communicated by W. J. Henwood, F.R.S.

† This mine was ultimately abandoned, about the year 1858, on account of its poverty.

Beneath the canga is the first jacotinga formation, about 60 feet in thickness, containing veins rich in the precious metal; the jacotinga partakes more of the character of mica slate, than of iron sand, and the auriferous veins in it are more like quartz than ironstone. This rests on a stratum of hard ironstone about three feet in thickness, which is the second jacotinga formation, but quartz is the predominating constituent, and rock is, according to Captain Treloar, a more correct name for it than jacotinga. This lode averages about four feet wide; it opens and contracts, and where it expands it is generally found most productive. Subjacent to the latter, is a layer of hard clay and mica slate, of about five feet in thickness, and then comes the rock formation, which has yielded the chief returns of gold. In the present workings, it is about ten feet wide; but in its longitudinal course, it so expands and contracts as to become in some places extinct. Where it expands, the dip is northerly.

Besides these, there is another lode in the hill on the opposite side of the valley, at a place called Maquine. Of this lode but little is known; a very large stream of water is issuing from it, and reports say that large quantities of gold, and some nuggets, have been found at the foot of the hill. The general direction of the lodes at Morro de Santa Anna may be said to be easterly and westerly, and their underlie northerly, but both vary, owing to the lodes hugging the contour of the mountain. The late proprietors worked only on a small scale, and on account of their want of knowledge of mining, chiefly confined their operations, during the last three years, to one *canudo*. The first of these three years was mainly devoted to clearing the workings, shifting stamping mill, and other preliminary operations, but even during this period the chief proprietor is said to have made a profit of £7,600.

The Rossa Grande Gold Mining Company.—This Company was formed in 1864, to work an extensive mineral property, called Rossa Grande, in which are said to be several gold mines. The estate is situated in the province of Minas Geraes, in the vicinity of the mine of Morro Velho. The city of Sabara and several villages are within easy walking distances of the property, and the road from Gongo Soco to Sabara and the St. John d'El Rey Company's mines passes through it. It is of great extent, and is estimated to comprise an area exceeding twenty-one square miles, or over 13,000 acres. The climate is said to be salubrious, and the characteristics of the lodes similar to those of the Morro Velho Mine.

The estate contains three distinct auriferous rock formations, which can be traced for miles, besides a jacotinga formation in the direction of Gongo Soco. Diamonds are said to have been found, and the alluvial deposit in the valley is believed to contain sufficient gold to make it remunerative for working. The first rock formation, or upper lode, is about six feet wide. It consists chiefly of white quartz and iron, and has yielded from four to forty oitavas, or from half an ounce to five ounces of gold per ton.

The second rock formation, or middle lode, varies in size from six to twelve feet. It is composed chiefly of quartz and auriferous arsenical pyrites. Lumps of gold have been found in it, and the ore in the swells has sometimes afforded fifty oitavas or upwards of six ounces of gold per ton. The third, or lower rock formation, is of greater magnitude than the other two, being about thirty-six feet wide. Its composition is mainly quartz and brown oxide of iron. The whole mass of this lode is said to be auriferous, and portions of it have yielded fifty oitavas of gold per ton. This mine has not yet produced any considerable amount of gold: the greater portion of the time which has elapsed since the formation of the Company, has been occupied in clearing out the old workings, and the erection of machinery.

East d'El Rey Company.—The Morro Saõ Vicente Mine, worked by this Company since 1863, is situated about twenty-four miles eastward of the Morro Velho Mine, and about twenty-two miles from Ouro Preto, the capital of the province of Minas. The highway to the latter from the interior passes through the estate, and the distance from the Emily Mines, formerly worked by this Company, is about seventeen miles. The estate contains several lodes, but the Champion Lode is considered to be more important than the others. The mine is on this lode, which runs obliquely to the cleavage planes of the containing rock; its course being S. 50° E., and underlie north, at an angle of 45°.

This vein belongs to the class of auriferous rock formations. It chiefly consists of quartz, pyrites, galena, tellurium, and gold; it can be traced for some miles, and its width varies from twelve to thirty feet. The rich shoots dip eastward, at an angle of 40°; the containing rock is clay slate. The mine is situated in a deep hollow, near the western boundary of the property, and, consequently, does not admit of drainage by an adit; pumping machinery, therefore, is used for this purpose. The excavation has gone down at a less slope than the dip of the lode, and, consequently, it has passed from one layer of the vein into another above it, and, since the returns of gold have been much higher in some months than during others, it follows that they vary in richness; yet, on the whole, the mine has been generally found to improve in descending. Its present depth is about 100 fathoms. This property has, until very recently, made regular returns of gold; but having, in common with other mines in the district, suffered much from want of workmen, who, in consequence of the war in which Brazil is now engaged, have been draughted for military service, it has been obliged temporarily to suspend operations.

The value of the gold remitted to England during the time the Emily Mine was being worked by this Company, amounted to 5,306*l.* 15*s.* 5*d.*

From the Morro Saõ Vicente:

	£	*s.*	*d.*
In 1864	2,852	7	4
1865, to May 1866	8,588	17	6
Total	£11,441	4	10

Brazil afforded its largest yield of gold about the middle of the eighteenth century, and from 1752 to 1761 the amount on which the royal fifth was paid, varied from 17,000 to 21,500 lbs. yearly. The production from that time fell gradually off, until in 1822 it was less than 1,000 lbs. From 1810 to 1817, the mean annual production of Minas Geraes, by far the most productive mining district of the country, is given by Humboldt as having been 4,288 lbs.

The gold at present produced in Brazil is almost exclusively obtained by the various large English companies, and particularly that of the St. John d'El Rey, from deep mining.

New Granada.—From the year 1819 to 1831, the ancient viceroyalty of New Granada was united with Venezuela, and during a portion of the time with Equador, forming the republic of Columbia, but is now, in common with each of the other States, nominally an independent republic. The mines of New Granada have been well described by Boussingault, Chevalier, and others, but we are without much recent information on this subject. The principal washings are situated in the provinces of Antioquia and Veraguas, in the former of which the detritus of all the rivers is said to be auriferous; and some quartz veins occurring in granite, and containing iron pyrites, are worked on a limited scale. The principal mines of gold quartz worked in 1850 were on the river Porce, the veins resembling in every respect the quartz lodes of other auriferous regions, and containing a considerable amount of iron pyrites, and other sulphides. Quartz veins are also numerous in the provinces of Panama and Veraguas, but the amount of gold contained in them is usually small. Chevalier estimated the yield of the Province of Antioquia for the year 1847–8 at 12,500 lbs. Mr. Danson,* from information based on the returns of the British Consuls, estimates the total amount produced from 1804 to 1848 at 40,817,066*l*. There is abundant evidence, however, that New Granada is rich in gold; and with a healthy climate, more energetic inhabitants, and a better government, there is no doubt that it would have annually produced large quantities of the precious metal. Various English Companies have at different times worked mines in New Granada, and even within the last twelve months discoveries were made of placer diggings, which attracted the general attention of Californian miners, and caused them to immigrate to the country in considerable numbers.

We believe, however, that the results obtained were not generally satisfactory, and that the diggings were very unhealthy.

Mariquita and New Granada Mining Company.—The most important gold mines belonging to British capitalists now worked in New Granada are those of Marmato, which are situated in the province of Antioquia. Previous to 1852, these mines were worked by an independent company, but in that year the above association was formed, for the double purpose of working the Marmato gold mine and the Santa Ana silver mine, both in New Granada.

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* Journal Stat. Society of London, xiv. 40, quoted by Whitney.

At the time of their purchase by the present proprietors, the mines of Marmato were provided with twelve stamping mills, representing, in the aggregate, 110 heads; which, during the year 1851, had crushed 12,488 tons of stuff, yielding on an average 11 dwt. 11 gr. of fine gold per ton, and resulting in a net profit of 8,343*l.* 6*s.* 8*d.* The Marmato mines are worked on deposits of auriferous pyrites, usually yielding a little more than half an ounce of gold per ton. The quantity of ore raised during the year ending March 1853, was 15,056 tons, and the quantity stamped 19,069* tons, the total produce being, gold, 10,711 oz.; silver, 5,988 oz., leaving a profit of 11,449*l.* 10*s.* 2*d.*

During the financial year ending March 1854, the ores raised amounted to 17,372 tons, and the quantity stamped to 18,225 tons, yielding gold, 10,170 oz.; silver, 5,895 oz., and resulting in a net profit of 9,767*l.* 13*s.* 9*d.* From March 1854 to 1855, the ores raised amounted to 14,154 tons, and the ores stamped to 18,288 tons, which produced 6,608 oz. of fine gold, and 4,193 oz. of silver; the profit for the year being 3,932*l.* 5*s.* 7*d.* In the following year, the ores raised amounted to 15,966 tons, and the quantity stamped to 17,668 tons, yielding 6,408 oz. fine gold; and 4,193 oz. of silver, and leaving a profit of 555*l.* 14*s.* 2*d.* on the operations of the twelve months. During the year ending March 1857, the ore stamped amounted to 19,370 tons, yielding 5,635 oz. of gold and 3,353 oz. of silver, leaving a profit of 2,184*l.* 19*s.* From this period to March 1858, 14,820 tons of ore were crushed, and 4,743 oz. of gold and 2,852 oz. of silver obtained; but resulting in a profit of only 541*l.* 0*s.* 8*d.*, as a large amount of machinery was erected in the course of this year. In 1859 the profits were 1,918*l.* 12*s.* 5*d.*, the stuff crushed, 19,598 tons, the gold obtained, 6,476 oz., and the silver, 3,874 oz. The profits in 1860 amounted to 2,778*l.* 8*s.* 4*d.*, but we have no data relative to the amount of ore treated, &c. For the year ending March 1861, the profit was 1,083*l.* 18*s.* 5*d.*; the number of tons crushed, 19,431 the yield of gold, 5,059 oz., and of silver, 3,150 oz. From the 28th February, 1861, to 24th of January, 1862, the operations of the mines were much interfered with by the Revolutionary War: only 16,859 tons of rock were crushed, and 2,592 oz. of gold, and 1,663 oz. of silver obtained. The loss on the operations of this year amounted to 2,745*l.* 15*s.* 9*d.*, and they were afterwards for some time almost suspended.

After the termination of the revolutionary movement, the works were again resumed, and between 31st March, 1864, and 31st March, 1865, 10,283 tons of stuff were crushed, and about 3,000 oz. of gold and 1,800 oz. of silver obtained, but the returns were not sufficient to meet the outlay. The Marmato property is at the present time being worked at a profit, and the Company has recently acquired the Aguacatal mines, in the immediate neighbourhood, from which satisfactory results are anticipated.

Peru.—Gold is found in many of the mountain passes, and nearly all its rivers wash down auriferous sands. Some of the richest gold diggings of the country are about Huaylas and Tarma. It is difficult to estimate with any degree of exactitude the amount of gold annually obtained in this country, the business of gold washing being carried on almost wholly by the Indians; but it is believed that the yearly production is somewhere about 2,400 lbs. Troy. In the valley of the river Chuquiaguillo the alluviums appear to be due to the abrasion of Silurian rocks, and are stated by Forbes to be eminently auriferous.* These have been worked from a very remote period, and the great quantities of gold found in Peru at the time of the Spanish conquest, had been, in great part, if not entirely, derived from detrital accumulations.

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* The number of tons crushed is always in excess of the weight of ore raised, as some of it is stamped more than once.

The rabbit-like burrows made by the Indian gold miners into the more auriferous beds, are still everywhere visible along the sides of valleys, where a supply of water was not too distant to admit of their transporting the pay dirt for the purpose of being washed; and later explorations have not unfrequently disclosed the mummies or skeletons of Indians who have, in these narrow and tortuous holes, perished from the falling in of the superincumbent earth, and have been buried along with their tools and mining implements. The gold washings in the valley of the Chuquiaguillo are at present conducted in the following way. The valley is, in the first instance, completely closed up, and the course of the river stopped by a rude wall or dam of stones and earth, provided with sluices, and having a portion of the wall somewhat lower, in order to serve as an overflow for any excess of water which may be collected. A longitudinal excavation is then made, close up to one side of the valley, of such a breadth as can be conveniently carried on by the number of hands employed on the works, and any stones or boulders that may be met with, too heavy to be carried off by the rush of water, are piled on one side, whilst the earth, sand clay, and gravel are removed by the force of the stream. On arriving at the several successive auriferous beds, the positions of which are known beforehand, from the results of previous trials, more care is taken in washing; but the whole of the pay dirt is flushed off, and, being heavier than the other constituents of the bank, deposits itself at a short distance from the workings. Here it is collected, and subjected to repeated washings in wooden troughs, until nothing but the grains of metallic gold remain behind.

The excavation is in this way gradually deepened, until the lowest available auriferous stratum has been reached, when the operations are abandoned, in order to commence similar washings on a line parallel with the first opening. Any stones or boulders met with in the new workings are thrown into the old one, and the excavation is thus carried directly across the valley.

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* Report on the Geology of South America, by David Forbes, F.R.S., communicated to the Geological Society, November 21st, 1860.

BOLIVIA.—Gold is found in many of the streams that flow down the eastern side of the cordillera, and at Choquecameta, near Cochabamba, at the sources of the Rio Grande, as well as at Tipoani, near Sorata; and in some other localities, the washing of auriferous sands is still carried on with profit. The gold-bearing strata occurring in the detrital accumulations of Bolivia, are generally known by the name of *veneros*, being, as it were, floors, or clay bottoms, on which gold had become deposited previous to its being covered by alternating beds of sand, gravel, and boulders. Above the first of these there are sometimes one or more similar deposits, again covered by layers of sand and gravel. The diggings are entirely confined to the sides of valleys and the beds of rivers, which afford the necessary supply of water for washing.

The celebrated washings of Tipoani and those of Yungas appear to belong to diluvial accumulations of the same geological age. There are no means of ascertaining the amount of gold annually obtained from the various mining districts of Bolivia, but the total produce is probably about 1,600 lbs. Troy.

Chili annually affords a considerable amount of gold, but we are without any reliable information of a recent date relative to the gold mines of that country. Humboldt estimated its yearly produce, in 1800, at 7,500 lbs. Troy.

CHAPTER VI.

BRITISH POSSESSIONS.—NORTH AMERICAN COLONIES.

DISCOVERY OF GOLD IN NOVA SCOTIA—GEOLOGY OF GOLD REGION—CORRUGATED QUARTZ AT WAVERLY—IMPRACTICABLE MINING LAWS—STATISTICS OF GOLD OBTAINED—CANADIAN GOLD FIELDS—PRINCIPAL GOLD WASHINGS IN CANADA—REPORT OF PARLIAMENTARY COMMITTEE—DISCOVERY OF GOLD IN BRITISH COLUMBIA—EXTENT OF THE GOLD FIELDS—GOLD CHIEFLY OBTAINED FROM ALLUVIAL DIGGINGS.

NOVA SCOTIA.—The whole of the Atlantic shore of this Province is bordered in an unbroken line by strata of a metamorphic character and probably of considerable geological antiquity, frequently broken through by eruptive rocks. These form a coast, in some places low and rugged, and in others boldly undulating; the soil is generally rocky and sterile, although there are large tracts well covered with timber, and affording prosperous agricultural settlements. Along the Atlantic shore, this district is generally low, gradually rising to a height of some three hundred feet, as it advances northward. Its coast line has, according to Dr. Dawson, a general direction of south 68° west, whilst its inland boundary, although presenting some considerable undulations, has a direction of south 80° west. The extreme breadth of this band, at Cape Canso, its northern extremity, is about eight miles; whilst, in its extension westward, it gradually increases, until at the west branch of St. Mary's River, eighty miles west of Cape Canso, it is known to be thirty miles wide. In the western counties its width has not yet been accurately ascertained, but here its entire breadth cannot be far short of fifty miles. Its total length corresponds with that of the peninsula of Nova Scotia. This band, in which almost the whole of the gold discovered has been found, chiefly consists of thick bands of slate and quartzite, highly inclined, and having a general north-east and south-west strike. In different localities these rocks, which probably belong to the Silurian epoch, have been penetrated by masses of granite, and in their vicinity the quartzites and clay slates usually present a highly metamorphosed appearance.

Since the gold discoveries in California and Australia have been generally known, and public attention has been directed to the conditions under which deposits of the precious metal usually occur, reports of similar discoveries have, from time to time, locally arisen in different parts of Nova Scotia. In every instance, however, either mica or iron pyrites would appear to have been mistaken for gold. Some years since, also, a considerable excitement was caused by an article in *Blackwood's Magazine*, in which it was affirmed that gold would be found in the hills to the south of Annapolis, and comparisons were instituted between that locality and the Valley of the Sacramento. Many persons were induced by this article to leave their ordinary occupations to seek for gold, but their researches having in all cases proved unsuccessful, the fever gradually subsided, and the subject was ultimately forgotten. It is also worthy of remark, that Dr. Dawson, so long ago as 1855, when describing the great metamorphic band, observes, "Quartz veins, however, occur abundantly in some parts of this district, and it would not be wonderful if some of them should be found to be auriferous."*

There is, nevertheless, no authentic evidence of the discovery of the precious metal in the province previous to 1860, when some hundreds of persons, tempted by rumours of gold having been found, commenced exploring near the head waters of the Tangier river. The amount of gold obtained in this locality was, however, so small that the miners ultimately became discouraged, and the excitement gradually subsided. In the month of March, 1861, a man who was stooping to drink at a brook, observed a piece of gold among the pebbles at the bottom, and, having picked it up, searched and found more. This took place about a mile to the east of the Tangier river. From this date attention became directed to the locality, numerous claims were taken up, and considerable quantities of gold were obtained by breaking the quartz with hammers, and washing the resulting dust in tin pans.

In June, the discovery of gold was reported near Lunenburg, at a locality called the Ovens. The veins at this place, although generally small, are frequently highly auriferous, and appear to cross each other in almost all directions in a metamorphic slate belonging to the great southern band. On these discoveries being made known, numerous claims were immediately taken up, and various small Companies formed for working the veins, presenting themselves numerously in the cliff.

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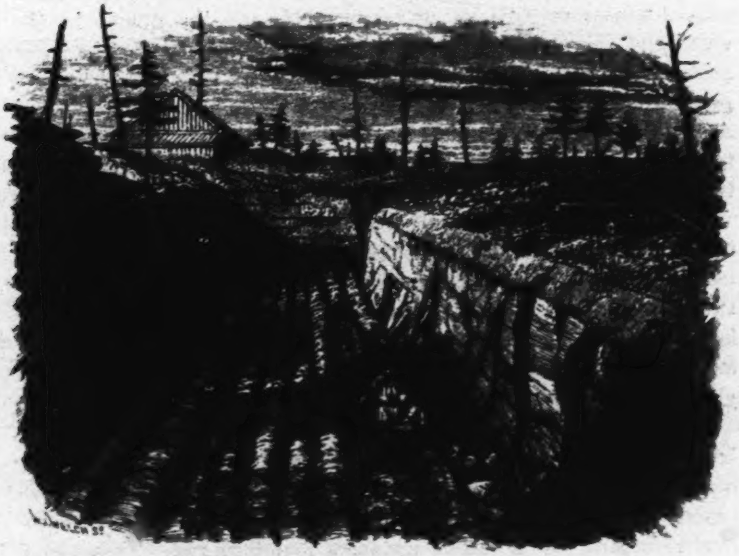
* Acadian Geology, p. 362.

Shortly after the discovery of the auriferous nature of the quartz veins, it was found that the sands on the beach beneath the headland also contained large quantities of gold; here claims were likewise rapidly staked off, and worked by means of cradles, so that the aggregate daily yield from the several shore operations soon reached one hundred ounces. Gold discoveries subsequently followed each other in rapid succession, at Lawrencetown, Dartmouth, Sheet Harbour, Isaac's Harbour, Sherbrooke, Waverly, and Oldham.

The most remarkable deposit of auriferous quartz hitherto found in Nova Scotia is undoubtedly that on Laidlaw's Farm, at Waverly.

Fig. 4.



CORRUGATED QUARTZ, LAIDLAW'S FARM, WAVERLY.
(*From a Photograph.*)

The principal workings are here situated near the summit of a hill composed of hard metamorphic shales, where openings have been made to the depth of some ten or twelve feet upon a nearly horizontal bed of corrugated quartz, of from eight to ten inches in thickness. This auriferous deposit, which is represented Fig. 4, is entirely different from anything we have seen elsewhere, and when laid open, presented the appearance of trees or logs of wood laid together, side by side, after the manner of an American corduroy road.

From this circumstance, the miners have applied the name of "barrel quartz" to the formation, which in many cases presents an appearance not unlike a series of small casks laid together, side by side, and end to end. The rock covering this remarkable horizontal deposit is exceedingly hard, but beneath it, for some little distance, it is softer, and somewhat less fissile. The quartz is itself foliated parallel to the lines of curvature, and exhibits a tendency to break in accordance with these striæ. The headings, and particularly the upper surfaces of the corrugations, are generally covered by a thin, bark-like coating of brown oxide of iron, which is frequently seen to enclose numerous particles of coarse gold, and the quartz in the vicinity of this oxide of iron is itself highly auriferous. The other gold veins of the Province present, generally speaking, few distinctive peculiarities, and very closely resemble those found in California and Australia. Their general course is north 60° west, and their dip towards the south-west, but there are not unfrequent exceptions to this rule.

In addition to gold, the most auriferous veins of Nova Scotia contain variable quantities of iron pyrites, mispickel, galena, blende, and, less frequently, a small proportion of argentiferous and auriferous sulphide of copper. Here, as elsewhere, the presence of the sulphides is regarded as a favourable indication of the richness of a gold vein; and a lead, containing much disseminated galena, almost invariably yields a certain quantity of gold.

The productive veins hitherto discovered have, as before stated, been found in the older rocks on the Atlantic shore, and commonly occur in parallel groups, near the centre of which, and parallel to the productive veins, a large reef of crystallised and comparatively unproductive quartz, is in many instances found to run. These large courses are locally called "bull veins," and usually contain small quantities only of the precious metal. The attention of the Nova Scotia gold miner has, contrary to the usual practice, been almost entirely directed to the exploration of veins of gold quartz, and alluvial digging has consequently been almost entirely neglected. There is, however, reason to believe that a careful examination of the alluvial deposits might lead to the discovery of gold. The thickness of the auriferous veins of this colony is less than those of California and some other countries, but they often contain visible gold.

In 1861, considerable excitement prevailed in the Colony with regard to the gold discoveries; and portions of the country, particularly those in the vicinity of Lunenburg, Tangier, Waverly, Lawrencetown, Sherbrooke, Sheet Harbour, and Isaac's Harbour, were staked off into claims, and more or less worked at a very shallow depth. Several Companies were also established with a view to quartz mining on a more extensive scale; but although many of the veins were found sufficiently auriferous to have warranted more extensive and systematic exploration, the larger Companies were not generally successful. This result appears to have been, in a great measure, the effect of the injudicious nature of the mining laws framed by the local Government, which limited the size of the claims to an exceedingly small area; and a somewhat heavy tax charged on each, rendered the prosecution of deep mining all but impossible. Nearly every man in the Colony thus became the possessor of one or more claims, which could only be taken up in accordance with certain maps prepared by the County Surveyor, by whom each district was laid out in parallelograms almost irrespective of the nature and position of the leads; and consequently, the larger Companies, if desirous to obtain a mining field of suitable extent, had to buy out, at a large price, numerous small claim-owners, who, although they could not work their several minute holdings, did not scruple to demand for them an exorbitant sum of money. It also not unfrequently happened that some claim-owner, whose fragmentary holding was so situated as to render it a necessity for the efficient working of a large location, took advantage of his position to refuse to sell except for an extortionate amount.

It is evident that such a system was totally incompatible with the rapid development of the mineral resources of the Colony, which, as a natural consequence, were most unfavourably affected thereby; but when these obstacles shall have been gradually removed, and reasonable laws have been enacted, there appears to be no reason for believing that gold mining will not become one of the profitable and lasting industries of Nova Scotia. The beach diggings near Lunenburg, which were at one time the most productive, have long since been, to a great extent, exhausted; but many of the associations organised for the purpose of mining and reducing auriferous quartz, still continue their operations, although, it is believed, with somewhat variable success.

The following are the official returns of gold obtained from Nova Scotia:—

PERIOD.	Average No. of Men employed.	No. of Crushing Mills.	Steam Power.	Water Power.	Quartz, Sand, and Gravel crushed.			Yield per Ton.			Gold from Alluvial Mines.			Total Yield of Gold.		
					Tons.	cwt.	lbs.	oz.	dwt.	gr.	oz.	dwt.	gr.	oz.	dwt.	gr.
Year ending Dec. 31, 1862 . . .	484	30	18	12	6,401	0	0	1	1	1	311	0	0	7,275	0	0
„ „ „ 1863 . . .	677	35	25	10	17,001	14	15	0	16	12	28	0	0	14,001	14	17
Nine Months ending Sept. 30, 1864	680	35	23	12	15,816	14	6	0	19	0	28	11	3	14,565	9	8
Year „ „ 1865	692	33	23	10	23,835	11	0	1	0	21	141	0	7	24,867	5	22

Canada.—The existence of gold in Canada first attracted public attention in 1847, although it is stated that M. de Léry, a French Canadian, had found specimens of this metal long previous to that date. In 1850, a considerable amount of local excitement was caused by the discovery of alluvial gold in the detritus of the Chaudière and various neighbouring streams; and an association, under the title of the "Chaudière Gold Mining Company," was formed for the purpose of washing gold in that district. In 1851, specimens of gold were shown in the Great Exhibition of that year, by Sir W. Logan, the Government Geologist, and the then recently formed Chaudière Company. In addition to the Chaudière, other localities have afforded this metal; among these may be mentioned St. Francis Beauce, Aubert-Gallion, Sherbrooke, and Melbourne. The auriferous region, according to Sir W. Logan, covers an area of from 3,000 to 4,000 square miles, and appears to occupy a great part of that portion of the province lying on the north-eastern side of the prolongation of the Green Mountains into Canada, extending up to the boundary line separating that colony from the United States. The greater portion of the gold hitherto obtained has been procured from superficial washings, none of the quartz veins which occur in the district having been extensively worked. The auriferous detritus frequently contains shells, and bones of animals of existing species.

The coarser materials of the drift are chiefly fragments and rounded pebbles of the clay slates and grey sandstone on which it lies; but these are more or less mixed with pebbles and boulders of serpentine and talcose slate. Sands containing specular, magnetic, and chromic iron ores, are also associated with this drift, as well as occasional rolled masses of white quartz, derived from the mountain chain bordering the district towards the north. The principal gold washings have been carried on on the Chaudière, Du Loup, and Touffe des Pins rivers, and the results obtained have been, on the whole, satisfactory. During the season of 1851-2, fifteen men collected about 80 ounces, and one nugget weighing two ounces was picked up. On the Touffe des Pins a lump of four ounces in weight was found.

From a personal inspection, made some years since, of the Canadian gold fields, we are inclined to believe that, in many localities, not only would the detritus, if worked on an extensive scale, pay a fair profit on the expenditure, but also that some of the quartz veins could be worked with considerable advantage.* Often, however, the thickness of the deposits is not sufficiently great, and the area locally occupied by the pay dirt is too limited to offer the requisite inducement for the installation of a regular system of sluicing.

On the 16th March, 1865, an elaborate report was made by a Parliamentary Committee, after a searching investigation of the subject, filling a closely-printed pamphlet of one hundred and twenty-six pages. A large number of witnesses, taken from all classes, were examined. Two or three extracts from this Report will show the character of the evidence elicited:—

"In regard to the extent to which gold has been mined or discovered, your Committee have obtained very valuable and reliable evidence. The Gold Mining Inspector, Major de Bellefeuille, states the whole quantity of gold produced during the past season, in the Chaudière Gold Mining Division, to be $116,000; and this, considering the comparatively small number of hands employed, must be regarded as a very handsome return, the average of the season being no less than $4 per day per man. The Gold Mining Inspector's Report, however, cannot be taken as representing the total quantity, as it comes only to the 30th of November, since which period mining operations have been constantly carried on, and with highly satisfactory results, on the Gilbert. The greater portion of the gold, so far obtained, has been taken from a small area on the Gilbert river, a few miles from its mouth, in the parish of St. Francis. It appears, however, that considerable quantities of gold were obtained in various parts of the country, of which he was furnished with no returns. Thus, for example, on the Stafford Brook, the evidence shows that $2,000 were obtained, while his return shows only $300.

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* In the report of a meeting of the present Chaudière Gold Mining Association, dated St. François, March 7th, 1866, accounts are given of fourteen different shafts now being sunk on quartz veins in the neighbourhood. One of these is twenty-four feet wide; another twenty; another thirty; and in neither have the walls yet been found. The assays reported run from $75 to $150 per ton; but these are evidently results obtained from picked samples. Thirteen cwt. of ore, taken from one vein, crushed, and worked in New York by mill process, yielded at the rate of $40 to the ton.

The whole expense of raising and working quartz at the mines, need not, it is said, exceed from $4 to $8 per ton.

Machinery is also reported to have arrived at Quebec, and only to await the opening of the season to be erected.

"In regard to the winter operations carried on, on the Gilbert, the evidence of the miners is highly important, as it was to some extent against their interest to give it. They have found hill tunnelling beneath the snow in winter, as profitable as alluvial washing in summer.

"With regard to the general richness of the Gilbert diggings, the evidence of these witnesses is conclusive. William Abbott took from one claim, of only sixty feet front, $1,750 in gold, during the early part of the summer; and in the month of August, he took from another claim, of only twenty-five feet frontage, $595. He had seen or found nuggets worth from $79 up to $300.

"John McCrea exhibited a nugget weighing 12 oz. 14 dwt., besides a quantity of gold in rough nuggets, amounting to many ounces in weight."

BRITISH COLUMBIA.—As early as June 1856, Mr. Douglas, the Governor of Vancouver's Island, reported to the Secretary of State the discovery of gold in British Territory, north of the 49th deg. of latitude, and stated that the earnings of the diggers ranged from 2*l.* to 8*l.* per day. In consequence, however, of the hostile attitude assumed by the natives, the number of diggers was very limited. Altogether, this discovery attracted at first less attention than might have been anticipated; but, in December 1857, Governor Douglas reported that the Indians themselves were extensively engaged in the search for gold, and that the accounts which had reached the neighbouring States of America had caused considerable excitement. It was not, however, until May 1858, that a stream of immigration, sufficient to overpower the opposition of the aborigines, had fairly set in, and the British public learnt for the first time, that the mainland of New Caledonia, as the district extending from the Red river to the Pacific was somewhat vaguely designated, was a rich auriferous country, which gave every promise of becoming a flourishing and important colony.

Beginning with the Fraser river, the main artery of the auriferous region, gold is known to exist, and has been worked, at a great many places on its course, from a point about forty-five miles from its mouth, up to near its source in the Rocky Mountains—in other words, from the 49th up to the 53d parallel of north latitude, a distance, taking in the windings, of some 800 miles. The south branch of the Fraser has its sources near Mount Brown, in the Rocky Mountains, in about 53° north latitude, 118° 40′ west longitude. Thence this branch flows for 290 miles to Fort George, a post of the Hudson's Bay Company. The north branch rises in an opposite direction. It receives its supply from a series of lakes lying between 54° and 55° of north latitude, longitude about 124° 50′ west, and runs a course of 260 miles to its junction with the south branch, some miles below the 54th parallel of north latitude. Here the union of the two branches forms the Fraser river proper. Adding the north branch, which is also a gold-bearing stream, the two will give a continuous stretch of auriferous territory, upwards of 1,000 miles in length, extending for many miles back into the country, but not including the tributary rivers which fall into the Fraser. In short, the river itself is known to be auriferous, and to pass through a gold-bearing country throughout its whole course. Gold is also found in most of the tributaries of the Fraser, of which no less than fifty-nine are known. The great length of the main river, and the number of its tributaries, will give some idea of the resources of the country for placer mining.

These facts do not, however, by any means convey an accurate view of the extent of the area of the gold fields, because these observations are limited to the central portions only of the country, while the whole of the upper portion of British Columbia is said to be auriferous.

Besides the gold found in the beds and on the shores of streams, the Fraser itself and many of its tributaries are skirted by terraces which yield gold. These terraces, or "benches," as the miners call them, run at intervals along both sides of the rivers for miles in length, and recede where the mountains fall back into the valleys, varying in breadth from a few yards to several miles. These are objects of curiosity, and add much to the beauty of the scenery in which they occur. They are generally found on both sides of the river, at the same place, sometimes at the same elevation on both sides, sometimes at different elevations, high on this, and low on the other side of the river, and in some places they are multiplied into several successive level plateaux, rising one above the other as they recede from the bank. These terraces are composed of the ordinary alluvial deposits, loam, gravel, stones, sand, and boulders, and are thick masses, generally rising to a height of from 150 to 200 feet.

There would also appear to be a considerable amount of gold in other localities besides the vicinity of the Fraser. Large yields have been obtained from the diggings between Fort Hope and Fort George, about 100 miles from its mouth. These mines are said to have yielded, during the season of 1861, an average of $17 to the hand, and a party of three men took, from three days' digging, $240. At Okanagan, sixty miles distant, the produce is stated to have been $4 to the hand. The Thompson river and its tributaries are also auriferous, and the North river has given good results.

Cariboo, however, appears to be one of the largest and richest gold districts hitherto discovered in British Columbia, and has produced large quantities of the precious metal. The gold regions of this colony appear to be a continuation, northward, of those of Oregon, and although it is difficult to procure any definite information on this subject, it is probable that they are of the same geological age as the great gold field of California. By far the largest proportion of the gold obtained from British Columbia is the produce of shallow placers, from which it is extracted by processes precisely similar to those employed in California and other auriferous countries. With regard to the quantities of gold annually furnished to commerce by this colony, there is some difficulty in arriving at correct figures. Nearly the whole amount obtained finds its way, by some means, to San Francisco; but, as a large proportion of it reaches that city through private hands, it renders impossible any attempt to obtain exact statistics. The total annual yield of British Columbia has been variously estimated at from 125,000 oz. to 150,000 oz.

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CHAPTER VII.

BRITISH POSSESSIONS.—AUSTRALIA AND NEW ZEALAND.

DISCOVERY OF GOLD IN AUSTRALIA—VICTORIA THE MOST PRODUCTIVE DIVISION OF THE AUSTRALIAN CONTINENT—GEOLOGY OF THE GOLD-BEARING ROCKS OF VICTORIA—QUARTZ VEINS—AURIFEROUS PLIOCENE GRAVELS—PRIMITIVE MINING—QUARTZ VEINS PRODUCTIVE AT ALL DEPTHS YET ATTAINED—PORT PHILLIP AND COLONIAL GOLD MINING COMPANY—YIELDS OF SOME OF THE PRINCIPAL MINES—TOTAL AMOUNT OF GOLD EXPORTED FROM VICTORIA—GOLD EXPORTED FROM NEW SOUTH WALES—SOUTH AUSTRALIA AND TASMANIA—QUEENSLAND—DISCOVERY OF GOLD IN NEW ZEALAND—PRINCIPAL GOLD FIELDS OF THAT COUNTRY—YIELD OF GOLD—ANNUAL PRODUCTION OF THE GOLD MINES OF THE WORLD.

AUSTRALIA.

THE Australian Cordillera has a direction nearly parallel with that of the coast line, from which it is usually distant from fifty to one hundred miles. This mountain range is principally composed of micaceous, argillaceous, and silicious shales, interlaminated with which the granite makes its appearance at the surface in numerous localities. The Australian slates often stand nearly vertical, and, like almost all other schistose rocks extending over large regions, have a general strike approximating to north and south. These have been much broken up and disordered by what are usually considered to be igneous rocks, such as porphyry, syenite, basalt and trap; and the general resemblance of the formation to that of the Uralian Cordilleras would appear to be so striking, that Sir R. Murchison, after an examination of the collections and maps of Count Strzelecki, in 1844, and without having visited the country, did not hesitate to express the opinion that this portion of Australia would, in all probability, be found to be auriferous. Specimens of gold had, however, been discovered by the Rev. W. B. Clarke as early as 1841, and in 1843 he first communicated the fact of the existence of gold in Australia to Sir George Gipps, then Governor of New South Wales, who enjoined him to silence, under the impression that a widely-spread knowledge of the discovery would be prejudicial to the general interests of the colony. The practical discovery of the presence of gold, in remunerative quantities, was, however, reserved to Mr. Hargreaves, a returned Californian, who was so much struck by the resemblance of the rocks of Australia to those found in the gold regions of California, that without waiting for the reply of Government to his notification that gold would be found there, commenced prospecting on the Macquarie river, New South Wales, and quickly found evidence of the existence of gold in considerable quantities.*

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* It may be well to insert here a certificate extracted from the Report of the Commissioners of the International Congress of Australian Statistics in 1861, which will probably be considered an unbiassed decision of the relative merits of the respective claimants to this discovery.

"95. *History of the Discovery of Gold in Australia.*—The first known discovery of the precious metal was made by Count Strzelecki in 1839, and was mentioned by him to some personal friends, and to Sir George Gipps, the then Governor of the colony of New South Wales. It was again discovered and specially noticed by the Rev. W. B. Clarke, of Sydney, in 1841. The attention of the colonial public, however, was not attracted to the subject until the existence of an extensive gold field throughout Australia was announced by Mr. E. H. Hargreaves in 1851. A long time previous to this announcement, namely, in 1844, and without being aware of the finding of specimens of the precious metal by Count Strzelecki and the Rev. W. B. Clarke, Sir R. Murchison publicly asserted the high probability of the existence of gold in Australia. This bold induction was based upon his knowledge of the geological formation of that country; and the wonderful results of gold mining in Victoria and New South Wales, afford a proof of scientific sagacity almost unparalleled in the history of science.

"James Macarthur,
"Edward Hamilton, } *New South Wales.*
"Stuart A. Donaldson,
"M. H. Marsh, *Queensland.*
"William Westgarth, *Victoria.*
"Edward Stephens, *South Australia.*
"Jas. A. Youl, *Tasmania.*
"J. E. Fitzgerald, *New Zealand.*

"Offices of the Congress,
Somerset House, London.
18th July, 1860."

The merit of scientific discovery is here awarded to Count Strzelecki. As to why he did not then make it known, we will allow the Count to speak for himself:—"I was warned," writes the Count, "of the responsibility I should incur if I gave publicity to the discovery, since, as the Governor argued, by proclaiming the colonies to be gold regions, the maintenance of discipline among 45,000 convicts, which New South Wales, Tasmania, and Norfolk Island contained, would become almost impossible; and unless the penal code should be amended at home, transportation would become a premium upon crime, and cease to be a punishment. These reasons of State policy had great weight with me, and I willingly deferred to the represen-

On the 8th of May, 1851, the Commissioner of Crown Lands received from him a notification that several ounces of gold had been collected on a branch of the Macquarie, and, shortly after, that a nugget weighing thirteen ounces had been discovered. It need scarcely be added, that the excitement at once became very great, and that all classes of the community left their usual occupations in order that they might rush to the gold diggings. On this discovery becoming an established fact, the Government laid claim to the lands, and began to grant licences, at the rate of thirty shillings per month, to dig for gold; and instituted, shortly afterwards, a geological survey of portions of the country.

VICTORIA.—This colony has hitherto produced a much larger amount of gold than any of the other divisions of the Australian Continent. Long before the appearance of any public announcement of the discovery of gold in Victoria, small pieces of this metal had been collected by shepherds and others, but no particular importance seems to have been attached to this fact, although well known to many of the older settlers, who appear to have made it the subject of conversation, and from whom rumours of its existence probably reached Europe. Mr. R. B. Smyth would appear to be strongly of this opinion, since, in his Paper on the "Mining and Statistics of Gold," page 101, Catalogue of the Victorian Exhibition of 1861, he remarks—"Indeed there is reason to believe, the statements of settlers who had returned to Europe in the early days of the colony, had left no doubt in the minds of scientific men at home that Victoria was a gold-producing country."* Gold is said to have been first found at Clunes in March 1850; and on the 10th June, 1851, it was discovered on a tributary of the river Loddon; on the 20th July at Mount Alexander; on the 8th August at Buninyong; and on the 8th September of the same year at Ballarat. Licences to dig were first issued on the 1st September, 1851, and such large yields of the precious metal were shortly after reported, that the colonists soon left their ordinary occupations to join in the exciting search for gold. The total male population of the colony was at that time only about 46,000, and the sudden withdrawal from their usual pursuits of nearly half this number at once produced a wonderful revolution

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tations of the Governor-General, notwithstanding that they were opposed to my private interests."—Quoted by Terry, "Thirty Years in New South Wales and Victoria." (London, 1863.)

* R. B. Smyth, Secretary of Mines for the Colony of Victoria.

in the social conditions of the country. The price of labour became enormously increased, all kinds of provisions rose rapidly in price, the value of property in Melbourne became seriously depreciated, and it was only after the arrival of a great influx of immigrants from Europe and from the adjacent colonies that society, to a certain extent, began to regain its equilibrium.

It would appear, from the investigations of Mr. A. R. C. Selwyn, the Government Geologist, that the age of the gold-bearing strata of Victoria is, geologically, much greater than that of the auriferous rocks of California, and further, that they belong to the lower palæozoic, or Silurian epoch, thus closely approximating in age and constitution to those of the great Uralian Cordillera.

In reference to this subject, we cannot do better than quote Mr. Selwyn's report on the "Geology of Victoria," which gives a concise and lucid description of the gold-bearing rocks of that colony:*—

"So far as at present known, the rocks of this period are the source whence the whole of the gold now produced in Victoria has originally been derived. They are exposed on the surface, at intervals, from the Grampians, on the west, to the extreme limits of the colony on the east. With a few local exceptions, they have a nearly true meridional strike or direction. Their great longitudinal extent is due to the crumpling and folding to which they have been subjected, causing the same beds to recur again and again at the surface, in a succession of synclinal and anticlinal undulations. Making due allowance for this great repetition of the same beds at the surface, the total vertical thickness of the series is probably not less than 35,000 feet.

"The lower members of the group consist chiefly of schistose and slaty rocks, with numerous beds of hard gritty quartzose and soft micaceous sandstones. Among the latter are beds that afford good freestone for building purposes, and, in the former, flags and roofing slates are occasionally met with.

"Various species of *Polyzoa* are the characteristic and most abundant fossils that have been found in the lower beds.

"In the upper portion of the series, which does not extend more than a few miles westward of the meridian of Melbourne, shaly 'mudstones,' associated with sandstones, very varied in colour and texture, are most prevalent. This portion is seldom affected by the true slaty cleavage, so characteristic of the lower beds, and it contains a rich assemblage of fossil animals, indicative of several of the subdivisions of the Upper Silurian period. The almost total absence of limestone bands, the number and extent of the quartz veins, and the constantly recurring protrusions, at short intervals, of granitic, and occasionally of plutonic trappean rocks, in dykes and large masses, are the most remarkable features in the physical structure of the lower palæozoic rocks in Victoria. The granite intrusions do not occur along any main axes of elevation, but are dotted about over every part of the area within which the palæozoic rocks are found. The stratified rocks, amongst which they have been intruded, are invariably hardened, and otherwise metamorphosed for a short distance from their junction. This alteration varies, in a very marked degree, with the mineral character of the altering mass; thus the change produced by diorites and feldspar porphyries is often very distinct from that produced by granite. Their intrusion appears very rarely, if ever, to have exercised any influence in determining the general strike, dip, and contortions of the palæozoic rocks; these invariably retain their general meridional direction, which is certainly remarkable, considering that the main water-shed, or axis of elevation, runs from east to west, and consequently almost at right angles to the strike of all the older rocks. It is difficult, under these circumstances, to understand what influence can have determined this feature in the physical geography of Victoria. That no great alteration, or even modification, of the water-shed has taken place, since the earliest tertiary periods, is well proved by circumstances connected with the physical geology of the formations of that period, as exhibited on either side of the dividing range.

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* Catalogue of the Victorian Exhibition of 1861, p. 177.

"The quartz veins occur throughout the lower palæozoic rocks, from the size of a thread to many feet in thickness. They have mostly a nearly true meridional direction, and are inclined either east or west, at angles varying from horizontal to vertical: occasionally they occur between the planes of the strata; more frequently in those of cleavage, and they often intersect both. They are true mineral lodes, and perfectly analogous in their mode of occurrence to all other mineral veins, whether of silver, lead, tin, copper, or any other crystalline mineral.

"The thickest and most persistent veins are found in the lower or older portions of the series, but the average yield of gold per ton is, however, greater from the generally thinner veins of the upper beds. These occur at Kilmore, Yea, Reedy Creek, Heathcote and Rushworth gold fields.

"The greatest depth to which any reef has yet (1861) been worked is about 460 feet. At this depth a yield of over five ounces per ton has been obtained.

"The total area occupied in Victoria by lower palæozoic formations with their associated plutonic rocks, inclusive of tracts in which the overlying tertiary deposits do not exceed 300 feet, cannot be estimated at less than 30,000 square miles."

It is, however, to be observed, that although the metamorphic rocks enclosing the auriferous quartz veins of Victoria appear to belong to a more ancient series than those in which gold quartz lodes are found in California, the drifts, constituting the deep diggings, seem, in both countries, to be almost, if not identically, of the same period, and are in each instance often capped by volcanic matter. In the report before referred to, Mr. Selwyn gives the following description of the auriferous tertiary gravels of Victoria:—

"The igneous rocks associated with them are strictly volcanic, and in no instance do they appear to be of older date than the close of the Miocene period. Their greatest development has taken place during the deposition of the Pliocene series, and in some instances it has evidently been continued to a period that could not be chronologically separated from the most recent geological events.

"The exact period in the tertiary epoch when the gold drifts commenced, is at present exceedingly doubtful. No beds are yet known in Victoria associated with, or forming a portion of such drifts, that contain fossil marine animals. Neither has any gold been obtained from beds below the known fossiliferous tertiary strata. The volcanic rocks, consisting chiefly of varieties of Trachytic Dolerites, Basalts, Trachytic Porphyries, &c., are in many districts interstratified, in contemporaneous layers, with the sands, clays, and gravels, of what are at present considered to be the oldest gold drifts, in which the lowest stratum, where the gold occurs, almost invariably consists of a water-worn quartz gravel. That there are gold drifts marking at least three distinct deposits, the results of successive upheavals and depressions, is quite certain; and it is now almost equally certain that the earliest of them was the result of the commencement of the oldest Pliocene period. In accordance with this view, they have been divided into Older Pliocene, Newer Pliocene, and Post-Pliocene deposits. These three stages sometimes occur in the same locality without the intervention of any volcanic rocks, in which case three bottoms or gold-bearing strata are found in one shaft, the last being always on the solid, unmoved palæozoic rock. About four hundred feet is the greatest known thickness of these Older Pliocene deposits including the associated volcanic rocks, and at this depth rich deposits of gold are found in them resting on the slopes, and in the hollows of what was once the old Pliocene sea bed. The exact relations of the gold-bearing drifts of the upper Tertiary periods to the marine Tertiary sands, clays, and limestones of the Miocene and Eocene series, is a very interesting point in Victorian geology not yet elucidated, and one which may have an important bearing on the probable extension of the deep gold leads of Ballarat and other gold fields.

"In following the leads they are invariably found to deepen in the general direction of the existing surface water-shed. Thus, at Ballarat and other gold fields on the south side of the dividing range, they deepen in a southerly direction; whilst at Clunes, Bendigo, &c., they invariably deepen in the opposite or northerly direction, and there seems no reason why they should not extend underneath a very large part of the extensive plains that stretch from the northern gold fields to the Murray, and from the southern flank of the dividing range to the sea-board, wherever the tertiary rocks forming these plains rest directly on the lower palæozoic strata."

It may here be observed that in Victoria there are, in addition to the great auriferous gravel beds generally referred to the Pliocene epoch, others of a comparatively non-auriferous character, which are found below them, and believed by Mr. Selwyn to belong to the Miocene period. In order to account for these older deposits being less auriferous than those occupying a higher position in the series, Mr. Selwyn has, we conceive, justly arrived at the conclusion, that there must be at least two distinct sets of quartz veins, that the older ones were formed prior to the Miocene period, and are comparatively barren, whilst the newer, formed after them, but before the Pliocene epoch, are productive. The former, therefore, will have furnished the materials for the barren Miocene gravels, whilst the disintegration of the latter has afforded the detritus forming the productive Pliocene deposits.* Below the basalt, the depth of auriferous alluvium rarely exceeds from ten to twenty feet, more frequently not so much, and in many places the basaltic rock is known to repose, for a considerable distance, directly on the bare bed rock.

In some parts of the country, not covered by basalt, such as Chinaman's Flat, Maryborough and Bendigo, the thickness of the alluvium is known to be as much as from 130 to 140 feet; and a still more

* Mr. Selwyn's Letter to the Victoria "Mining Record," February 22, 1866.

remarkable instance has been exposed, where a shaft has been sunk to a depth of more than 400 feet, passing, from top to bottom, through a conglomerate formed of water-worn pebbles, and blue clay. At this place, the shaft, although not passing through eruptive matter, is surrounded by a volcanic country, and, up to January 1865, had not yet bottomed on the bed rock.

In Australia, as in California, there are three distinct sources from which the supply of gold is directly obtained,—viz., shallow placers, deep diggings, and veins of auriferous quartz.*

In the early days of gold digging, the miner usually contented himself with excavating shallow pits in the clays and gravels, found in beds and gullies of the creeks, or with washing the soil from slopes of hills intersected by auriferous quartz veins.

The methods of extracting gold were in all cases exceedingly primitive, and conducted with the aid of the smallest possible amount of plant and materials. When a light sandy earth had to be dealt with, it was either washed in the tin pan, or passed through a cradle; whilst if, on the contrary, the soil was mixed with tenacious clay, it became necessary to puddle it before passing it through this simple machine. In order to effect this puddling, the auriferous material was thrown into a large tub, with a sufficient quantity of water, and continually stirred with a shovel until the clay became softened and held in suspension in water, which from time to time was run or drawn off, and a fresh supply added, until the sands and gravels remaining in the bottom had become sufficiently free of clay to admit of being cradled without difficulty. The residues, thrown away by the miners, who worked by this system, were necessarily often very rich, and, after exposure to the disintegrating action of the atmosphere, were frequently reworked for gold with considerable profit.

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* "Gold is now found to occur, not only in quartz veins and the alluvial deposits derived from these and the surrounding rocks, but also in the claystone itself, and, contrary to expectation, flat bands of auriferous quartz have been discovered in dykes of diorite which intersect the Upper Silurian or Lower Devonian rocks. Quartz of extraordinary richness has been obtained from these bands, and the new experience of the miner is leading him to look for gold in places heretofore entirely neglected. It is probable that some time may be lost, and that his labours may not always be well directed or successful, but it is commendable that he should not be deterred from exploration by warnings and remonstrances, founded on surmises often baseless. If he had closely followed the older precepts, we should, at this moment, have been dependent for our yield of gold on the shallower alluviums, and the surface only of the veins of quartz. The miner is, however, prospecting the deeper tertiaries with well-grounded hopes of success, and some of the shafts which have been sunk to penetrate the veins are as deep as 590 feet. From these levels very rich quartz has been obtained."—*R. B. Smyth, Intercolonial Exhibition*, 1866, p. 5.

After a time, the small pits sunk in valleys and the bottoms of creeks, were carried down to the bed rock, where it was discovered that the principal part of the gold had usually accumulated. From the bottom of these, small drifts were extended in all directions, and the wash dirt carefully collected from the face of the bed rock, and brought to the surface for the purpose of being washed. The claims originally granted to the Australian miner were exceedingly limited, the dimensions of each being in most instances 10 feet by 10 feet, or 16 feet by 8 feet, and consequently this method of mining was only sufficiently advanced for operations conducted on such a restricted scale. These workings, however, in many cases crushed together, before the whole, or even the largest portion of the gold-bearing stratum had been removed; and hence one of the principal reasons why, in nearly all the more important gold fields, the alluvium has been repeatedly worked over with profitable results. This method of mining is still extensively practised in Victoria, but the puddling is now, in the majority of cases, effected either by horse or steam power.

From the general scarcity of water in the auriferous districts of Australia, all operations connected with the process of gold washing require to be carried on with a view to the utmost economy of this indispensable agent, and consequently the hydraulic mining of California is almost unknown in the country, and even sluicing on a large scale is comparatively little practised. The deep diggings, on deposits of the Pliocene tertiary epoch, are in Victoria most frequently worked by means of shafts of greater or less depth, from which, when the bed rock has been reached, galleries are extended, and the pay dirt is removed, very much in the same way as coal is extracted from seams of that mineral.

The pay dirt, on reaching the surface, is subjected to the operation of puddling, and the residue remaining in the machine subsequently washed for the gold it contains. This method of carrying on operations is not only expensive, but the working of deep leads, as the beds of these ancient watercourses are called in Australia, is frequently much impeded by the influx of water, and a relatively large proportion of the steam-power employed in the gold fields is utilised in pumping water from deep workings.

The time necessary for sinking a shaft from the surface to the pay dirt may vary from a few months to one or more years, during the whole of which time no possible return can be made for the outlay expended; and it is only after reaching the "gutter," that the harvest of the miner begins to be reaped. From a report, in 1861, of Mr. Davidson, one of the mining surveyors of Ballarat, it appears that the average yield of the celebrated leads, named the Golden Point, Inkermann, Redan, and Nightingale, situated in his district, was from 10 dwt. to 2½ oz. per cubic yard, and that the thickness of the pay dirt varied from one to twelve feet. Another mining surveyor states, that at the Waterloo Company's claim, Ballarat, the total quantity of gold obtained was 6,750 oz., which, at 80*s.* per ounce, would amount to 27,000*l.* This company was occupied two years and one month in working out the claim, and the total expense of carrying out the works was 5,824*l.*, thus leaving a clear profit of 21,176*l.* as the result of the operations.

In some localities, as at Sandhurst, where the pay dirt is composed of water-worn pebbles, strongly bound together by ferruginous and silicious cements, it is crushed in an ordinary quartz mill, and the gold extracted by amalgamation in the usual way. The mining surveyors speak favourably in their reports of this system of treatment. These auriferous deposits of the Pliocene epoch are found at Ballarat, Smythesdale, Creswick, Raglan, Ararat, Sandhurst, Indigo, near Beechworth, Maryborough, and various other localities in the province.

Wherever the level of the ancient deposit is above that of the modern valleys, adit levels, or tunnels, are brought in on the leads, and the work is then conducted precisely as if it had been reached by the sinking of perpendicular shafts. This method of mining has, however, many advantages over deep sinking, both with regard to the drainage of the workings, and transport of the stuff to be washed, and is invariably adopted wherever the conformation of the district admits of its introduction. In order to avoid the expense and loss of time involved in the drainage of long levels from the bottom of the shafts, to reach the deposits of pay dirt, it is now usual to explore the ground by means of bore-holes: and the true position of the auriferous channel having been thus approximately ascertained, the shaft is subsequently sunk in such a position as most conveniently to command the lead, and reach it with the least possible time and expense. As far back as 1855, the attention of the local Government was directed to the necessity of supplying the gold fields with water, and in 1860 a first grant of 50,000*l.* was appropriated for that purpose this was followed, in 1861, by a further sum of 75,000*l.*, for the extension of similar works.

In the early days of the colony, the operations of quartz miners were exclusively confined to such outcrops as showed visible gold, and the rock being either broken by a hammer or pounded in a mortar, the gold was afterwards washed out in the ordinary tin pan. It was, however, soon found that the working of veins in depth could be prosecuted with advantage; and at the present time, from one-fourth to one-third of the gold obtained in Victoria is the result of treating auriferous quartz. Whenever metamorphic slates appear at the surface, quartz veins are generally found, the general direction of the greater number being nearly north and south. These veins vary in thickness, from a mere thread to forty or even fifty feet. The magnetic bearings of the northerly and southerly veins are, with rare exceptions, confined within a variation of some 24°, and the strike of the easterly and westerly lodes, which are much less numerous, is nearly at right angles to them.

The machinery usually employed in Australia for the treatment of auriferous quartz, is the ordinary stamping mill, which will be described in a subsequent chapter, each head, on an average, striking sixty blows per minute, and weighing seven cwt. About one horse power is consumed by each of these heads, a nominal ten-horse engine being most frequently attached to a battery of eight. The crushed quartz flowing from the stamping mill is conducted over blankets, mercurial riffles, amalgamated copper plates, and various arrangements similar to those employed in California and other gold-producing countries; the Australian miners experience the same difficulty, elsewhere complained of, in separating gold from iron pyrites, and other metallic sulphides. The miners generally have, in common with those of other countries, come to the conclusion, that better results, with regard to profit, are to be obtained by crushing large quantities of only moderately rich stuff, with powerful and well-arranged machinery, than from treating a limited amount of very rich rock on a small scale.

When, as at first, the cost of crushing and amalgamating a ton of quartz amounted to some 80*s.*, a very small proportion of the auriferous veins of the colony could be treated with advantage; but now that the total expense of raising and treating a ton of rock by steam-power has, under favourable circumstances, been reduced to about 16*s.* 4*d.*, there are numerous reefs throughout the gold-mining region affording satisfactory results, and many others that would do the same, if extensively and judiciously worked. Many of the reefs have yielded quartz of extraordinary richness, as, for instance, one at Castlemaine, from which 266 oz. per ton were obtained, although this cannot be regarded as the average produce of the vein.

Generally speaking, also, the yield of the quartz veins of Victoria has not been found to decrease in depth, and those which have been wrought below 500 feet from the surface have experienced no diminution in their produce. Quartz mining requires a considerable amount of capital, and is subject to all the usual fluctuations of mining enterprises, excepting, however, such as are influenced by the varying prices of metals; but, in spite of all its disadvantages, it has been found a highly remunerative occupation, and from the productiveness and extent of its reefs, it is certain that Victoria presents an attractive field for the investment of capital in this department of industry.

Two of the most productive gold fields of Victoria have been those of Ballarat and Bendigo. The gold field of Ballarat is the largest within the mining district of that name, and the centre of other outlying gold fields, among which are Clunes, Creswick, Smythe's Egerton, Gordon, Steiglitz, Linton's, Carnham, &c.

The town of Ballarat is, with perhaps the exception of Sandhurst, the chief town of the district in which Bendigo is situated; and Castlemaine, the chief town of the district of that name, the most important and largest within the six mining districts. The mining carried on in this district is, for the most part, of the deep alluvial class, the bed rock being seldom found at a less depth than 200 or 300 feet.

Bendigo, the largest auriferous field of the Sandhurst district, was originally a shallow gold field, some of its famous gullies, yielding many ounces to the tub, having been scarcely a yard in depth. The soil from these has now, however, been all cleared away to the bed rock, and passed through the puddling machines, from whence it has issued in the form of a pasty sludge which may be seen in all directions overflowing the surface of the county. In place of the placer diggings, which are at present, to a considerable extent, exhausted, the chief industry of Bendigo is now concentrated on its quartz mines, which extend over an area of some forty square miles. On the north side of the gold field, however, deep placer mining has been extensively carried on, where the bed rock was found to dip rapidly, and this extension of the Bendigo gold field is now being extensively worked.

It would be difficult to ascertain the average produce of the quartz crushed in the different reduction establishments in the colony since the commencement of quartz mining, but it appears from the Surveyor's reports for 1860, that, in that year, 61,075 tons of rock from the Ballarat district yielded 38,378 oz. 6 dwt. of gold = 12 dwt. 13 gr. per ton.

From the Beechworth mines 3,725 tons 16 cwt. of quartz gave 13,862 oz. 6 dwt. of gold = 3 oz. 14 dwt. 10 gr. per ton.

The Sandhurst district afforded 2,678 tons 15 cwt. of quartz, yielding 6,361 oz. 8 dwt. of gold = 2 oz. 7 dwt. 12 gr. per ton.

From the Maryborough district there were crushed 4,548 tons, producing 6,345 oz. 1 dwt. of gold = 1 oz. 7 dwt. 21 gr. per ton.

The district of Castlemaine produced 13,301 tons 15 cwt., affording 14,955 oz. 11 dwt. of gold = 1 oz. 2 dwt. 11 gr. per ton.

Ararat yielded 1,265 tons 10 cwt. of quartz, which gave 2,002 oz. 10 dwt. of gold = 1 oz. 11 dwt. 15 gr. per ton.

In the same year there were 294 steam-engines, of the aggregate horse-power of 4,137, employed in alluvial mining, and 417 engines of the aggregate power of 6,645 employed in quartz mining. There were, in addition, 138 water-wheels and 3,958 horse puddling machines in operation. The total approximate value of the mining plant in the colony was, at the same period, estimated at 1,299,303*l.*

The total number of steam-engines employed for gold-mining purposes at the end of 1864, was 888, of which 441 were used in alluvial, and 447 in quartz mining. In the following year there were 17,326 miners engaged in quartz mining, more than 2,000 distinct reefs had been discovered and named, and 491 steam-engines, equal to 8,606 horse-power, and 62 engines driven by water or horse power, were employed in quartz mines for crushing, winding, pumping, &c.

On the 31st December, 1865, the number of leases in force in the different districts were as follows :—Ballarat, 88; Sandhurst, 318; Maryborough, 228; Castlemaine, 85; Beechworth, 300; Ararat, 24.

There were 62,131 miners engaged in getting gold from the auriferous alluviums, and 4,131 machines of various kinds, 4,428 sluice boxes, and 648 sluices and toms used exclusively in this kind of mining. The quantities of gold obtained by the alluvial miners during three years were, as nearly as can be ascertained, as follows:—1863, 1,133,567 oz.; 1864, 1,041,830 oz.; 1865, 1,093,801 oz.*

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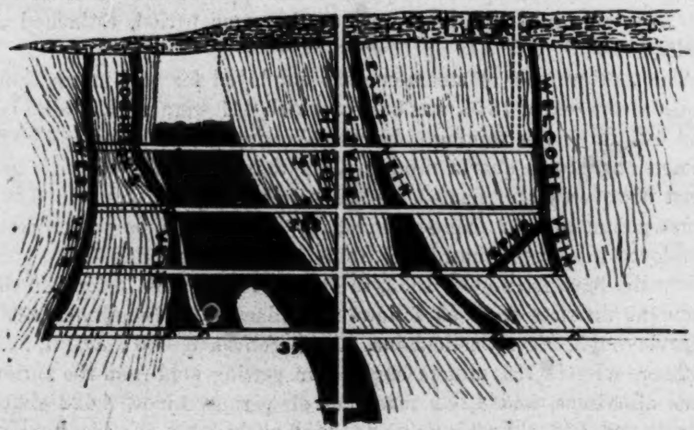
* R. B. Smyth, International Exhibition, 1866, p. 24.

Clunes.—*Port Phillip and Colonial Gold Mining Company.*—This is the most important and extensive quartz-mining enterprise in the colony. The extent of the claim is 160 acres, held on a lease for twenty-one years, from 1st January, 1857, at a royalty of 7½ per cent. on the gross value of the gold raised. Operations were commenced by the present Company in 1857, and in the following year, Mr. Selwyn, the Government Geologist, thus writes of its mines and establishments:—"Within the last twelve months, I have visited all the principal quartz reefs and crushing establishments on the northern and western gold fields, and the Port Phillip and Colonial, and Clunes Company, is the only one I have seen, of which it would be possible to say, that it leaves little to be desired, either as regards the system of working the mine, or the general arrangement and management of the machinery."

The reefs worked are five in number, and are enclosed in a soft white and brown slaty sandstone, which, like the veins themselves, runs nearly north and south, and generally dips towards the east. With the exception of the overlying tabular basalt, no igneous rocks occur on the surface within three miles of these veins. The nearest granite is at Mount Beckworth, nine miles west of the Company's property. Nearly three-fourths of the land held by the Company is occupied on the surface by a stratum of tabular basalt, and it is only in the south-western corner, where this rock has been removed by denudation, that the shales and sandstones, with their associated quartz veins, come to the surface.

The following section, at the northern extremity of the Company's property, Fig. 5, will serve to give a general idea of the formation of the county, and the relative positions of the different veins worked.

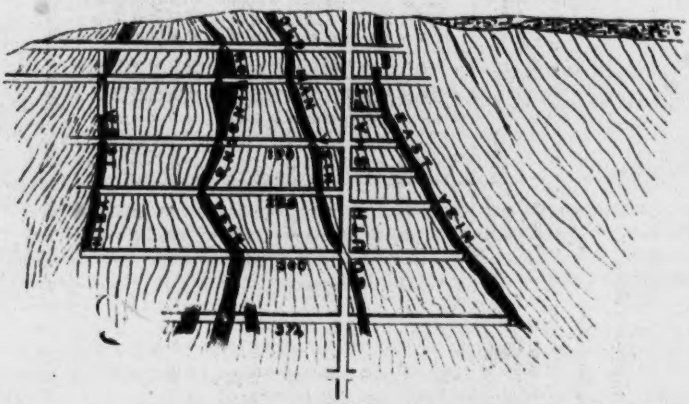
Fig. 5.



Clunes.
(*North Cross Section.*)

The extreme irregularity in the width of these veins in different portions of their course, will be at once perceived by comparing the cross section at the southern boundary of the property, Fig. 6, with the foregoing. It will be remarked, that in this portion of the ground, the "Old Man Vein" is very much contracted, whilst the "Welcome Vein" has become entirely obliterated. This alternate swelling and pinching out of quartz veins appears to be peculiarly characteristic of lodes of auriferous quartz in all parts of the world.

FIG. 6.



CLUNES.
(*South Cross Section.*)

The greatest depth to which this mine has been worked is 485 feet, and the total amount of quartz treated since the commencement of operations in 1857, to October 1866, has been 319,695 tons, yielding 185,488 oz. 0 dwt. 8 gr. of gold, giving an average produce of 11 dwt. 14½ gr. per ton. We are indebted to Mr. C. H. Fielder, the secretary of the Company, for the information embodied in the following tables:—

TABLE I.

Return of Quartz crushed from June 1857 to October 1866.

Period.		Amount crushed.	Net Amount of Gold produced.			Average per Ton.			Royalty.			Total.		
		Tons.	oz.	dwt.	gr.	oz.	dwt.	gr.	£	s.	d.	£	s.	d.
1857 .	7 Months.	4,146½	6,780	14	8	1	12	17	2,650	11	1	26,764	4	6
1858 .	9 ,,	11,320½	15,764	9	0	1	7	20	6,130	2	2	61,301	3	5
1858-9 .	12 ,,	17,542	18,165	12	12	1	0	17	6,202	15	8	71,497	4	5
1859-60	12 ,,	21,694	17,466	16	9	...	16	0	5,135	14	4	68,476	5	8
1860-1	12 ,,	32,258	24,326	6	3	...	15	2	7,178	3	2	95,706	13	3
1861-2	12 ,,	34,236	22,312	0	17	...	12	20	6,479	18	4	86,395	12	11
1862-3	12 ,,	40,360	22,986	1	19	...	11	9	6,850	4	5	91,336	5	8
1863-4	12 ,,	44,149	17,611	8	0	...	8	0	5,227	1	9	69,694	7	2
1864-5	12 ,,	54,418	20,596	15	12	...	7	13	6,074	3	0	80,692	7	8
1865-6	12 ,,	59,576	19,775	16	0	...	6	15	5,893	16	10	78,584	19	1
		319,695	185,488	0	8	...	11	14½	57,522	10	9	730,454	3	9

The subjoined table (II.) gives the monthly results of the operations of the Company in Victoria, from January 1865 to June 1866, both inclusive. The crushing apparatus consists of seven stamping mills or batteries, composed in the aggregate of eighty heads, each weighing, including the stem, from 6 to 8 cwt.

TABLE II.

PORT PHILLIP AND COLONIAL GOLD MINING COMPANY.

Cost and Returns from January 1865 *to June* 1866.

PERIOD.	Quartz Stamped.	Quartz Stamped per Week.	Duty, per Stamp head per Day.		Produce of Gold.	Produce per Ton.		Gold per Ton in Tailings, by Assay.*		Total Mine Cost per Ton.		Total Reduction Cost per Ton.		Pyrites				Price of Gold per oz.			Proceeds of Gold.
														Treated.	Yield per Ton.		Profit.				
	Tons.	Tons.	cwt.	qr.	ounces.	dwt.	gr.	dwt.	gr.	s.	d.	s.	d.	Tons.	oz.	dwt.	£	£	s.	d.	£
1865.																					
January .	3,412	853	49	0	1,039	6	2	1	15	...		...		...	...	...	...	3	16	0	4,147
February .	4,551	910	48	2	1,566	6	22	1	15	...		...		...	...	...	...	3	16	0	6,272
March . .	3,677	919	54	0	1,006	5	11	1	22	15	0	7	9	85	2	7½	460	3	16	0	4,163
April . .	3,326	831	58	0	868	5	5	1	3	16	0	6	6	...	...	...	...	3	16	0	3,434
May . .	6,142	1,024	58	0	1,824	5	22	1	20	15	6	5	2	...	...	...	...	3	16	0	7,213
June . .	4,530	1,132	56	0	1,836	8	2	2	3	14	9	7	0	...	...	...	...	3	16	0	7,322
July . .	4,688	1,172	57	0	1,761	7	12	1	23	13	2	5	6	23	3	1½	208	3	16	0	7,026
August .	4,787	1,197	57	2	2,254	9	10	2	15	12	8	5	6	25	3	1	222	3	16	0	9,019
September	6,817	1,136	54	0	3,465	10	4	1	22	14	11	5	2	30	4	12	413	3	16	0	13,725
October .	4,774	1,193	56	3	1,997	8	8	2	3	11	2	7	3	...	...	...	...	3	15	4	7,876
November	4,676	1,169	57	3	1,661	7	2	2	15	14	7	8	2	...	...	...	...	3	15	1	6,562
December	4,195	1,049	54	3	1,316	6	7	2	6	16	1	11	2	...	...	...	...	3	14	10	5,172
1866.																					
January .	4,177	1,044	53	0	1,377	6	14	2	6	15	3	9	4	...	...	...	...	3	14	10	5,438
February .	4,511	1,128	53	2	1,602	7	2	1	22	13	11	8	0	105	3	1	856	3	14	10	6,319
March . .	4,830	1,207	57	0	1,514	6	6	1	23	11	9	6	7	...	...	...	...	3	15	4	5,999
April . .	6,778	1,129	55	0	1,946	5	18	2	6	10	10	6	6	40½	2	10	280	3	15	4	7,728
May . .	4,783	1,196	57	0	1,111	4	15½	1	16	11	0	6	2	19	2	11	...	3	15	9	4,435
June . .	4,877	1,219	63	3	911	3	17¾	1	17	10	6	7	0	19	3	3	286	3	15	9	3,616

* A portion of this gold is recovered by washing the tailings, and roasting and amalgamating the pyrites obtained.

From estimates very carefully made by the Colonial Secretary of Mines, which, however, are not given as being absolutely correct, it appears that the total quantities of gold obtained in Victoria from quartz veins were, during the three years undermentioned, as follows:—

	oz.	dwt.
1863	493,499	0
1864	503,618	5
1865	450,000	0

The following particulars relative to a few of the principal mining operations in the colony have been collected from the files of *Dicker's Mining Record.*

BALLARAT.—The following workings in this locality are classed as alluvial:—

Cosmopolitan.—Commenced 1857; claim 880 feet on the gutter; basaltic rock 112 feet in thickness; total length of drivages, 2,500 feet; stuff puddled and washed up daily. This undertaking had produced 38,249 oz. 4 dwt. 19 gr. of gold, of the value of 152,442*l.*, and paid dividends to the amount of 125,454*l.* up to November 19th, 1864.

United Extended Band of Hope.—Extent of claim, 36,040 feet; commenced in 1856, and had yielded gold to the value of 259,547*l.*; of which amount 147,200*l.* had been paid in dividends up to 31st December, 1865.

Defiance Company.—Commenced, 1856, and had afforded 20,197 oz. 18 dwt. of gold of the value of 81,102*l.*; out of which 66,483*l.* had been paid in dividends up to December 24th, 1864.

Koh-i-Noor.—Commenced, 1857; produce, 60,332 oz.; value, 241,233*l.*; dividends paid, 176,080*l.*, up to December 6th, 1864.

Prince of Wales.—Commenced, February 1857; gold produced, 22,912 oz. 11 dwt. 19 gr.; value, 90,735*l.*; dividends paid, 51,552*l.*, up to December 10th, 1864.

Alston and Weardale.—Commenced, July 1858; gold produced, 2,585 oz. 15 dwt. 3 gr.; value, 10,451*l.*; dividends paid, 4,884*l.*, up to August 1864.

Great Extended.—Commenced, 1857; amount of gold produced, 82,754 oz. 9 dwt. 14 gr.; value, 325,967*l.*; dividends paid, 274,450*l.*, up to December 4th, 1864.

BUNINYONG.—*Buninyong Mining Company.*—Alluvial. Commenced, November 1857; gold produced, 43,175 oz. 16 dwt. 22 gr.; value, 172,695*l.*; dividends paid, 88,007*l.*, previous to November 26th, 1864.

DAYLESFORD.—*New Wombat Hill.*—Alluvial; commenced April 1861; gold produced, 14,942 oz. 14 dwt. 19 gr.; value, 58,059*l.*; dividends paid, 38,400*l.* up to November 5th, 1864.

BENDIGO.—*Catherine Reef United.*—Quartz; claim, 501 yards, held under a lease from the Government; vein 4 feet wide, with strike 23° west of north, and a dip to the east; commenced, March 1861; gold produced, 24,930 oz. 15 dwt. 12 gr.; value, 95,784*l.*; dividends paid, 29,735*l.* previous to December 17th, 1864.

SCARSDALE.—*Avonclift.*—Alluvial; area of claim, 133 acres; depth of basalt, 84 feet; two gutters running parallel; depth of wash dirt, 4½ feet; gold produced, 4,272 oz. 14 dwt. 12 gr.; value, 17,090*l.*; dividends paid, 4,000*l.* up to September 1864.

HAPPY VALLEY.—*British Gold Mining Company.*—Alluvial; extent of claim, 50 acres; held under a lease from the Government; the gutter is 30 feet in width, and the depth of the wash dirt, about 3 feet; commenced in July 1860; produce of gold, 17,501 oz. 6 dwt. 1 gr.; value, 70,126*l.*; dividends paid, 40,800*l.* up to March 4th, 1866.

Try Again Gold Mining Company.—Alluvial; commenced, September 1859; extent of claim, 49 acres; held under a lease from the Crown; wash dirt from 2 to 3 feet in thickness; gold produced, 6,153 oz. 7 dwt. 8 gr.; value, 24,535*l.*; dividends paid, 9,802*l.* up to December 10th, 1864.

Cleft in the Rock.—Alluvial; commenced, November 1859; extent of concession, 50 acres. held under a lease from the Crown; main lead bearing north and south; thickness of pay dirt, from 2 to 7 feet; produce of gold, 3,848 oz. 14 dwt. 11 gr.; value, 15,287*l.*; dividends paid, 5,960*l.* up to December 12th, 1864.

WOOD'S POINT.—*Alps Great Central Gold Mining Company.*—Quartz; commenced, July 1863; produced 27,500 oz. of gold; value, 92,812*l.* up to July 1865.

Hope Mining Company.—Quartz; commenced, July 1863; produce of gold 8,732 oz. 3 dwt.; value, 27,032*l.* up to July 1865.

By an Act for the better management of the gold fields of Victoria passed by the Colonial Parliament in 1857, the whole of the auriferous region was divided into the six following mining districts, each called after its chief gold field, or rather after the chief town in its most important gold field:—Ballarat, Castlemaine, Sandhurst, Maryborough, Beechworth, and Ararat; but this division is entirely arbitrary, and totally unconnected with any geographical or geological features of the country. The only connecting link between the Colonial Government and the gold fields, is the Warden, whose duty it is to attend to the settlement of small disputes connected with the gold-mining interests; to report on the advisability of granting leases of mineral lands, and to draw up periodical reports relative to the state of the gold fields in his district. Each gold field, and sometimes each division of a gold field, when it is a large one, has its own Warden, whose correspondence with the Government passes through the hands of the Chief Warden. Each district has its own Mining Board, of which the members are chosen by ballot by the gold miners, and which arranges all gold-mining questions within its jurisdiction. Each member receives a small allowance for his attendance at the board, and holds office during three years, at the expiration of which he is eligible for re-election. A Court of Mines is also established within each mining district, before which come all disputes above the adjudication of the Chief Warden. The judges of these courts are independent of the Colonial Government, and hold office *quamdiu se bene gesserint.* From the decisions of the judges of the Court of Mines there is a power of appeal to the Supreme Court of the Colony.

The following amounts of gold have been exported from the Colony of Victoria since 1850:—

PERIOD.	QUANTITY.			VALUE.		
	oz.	dwt.	gr.	£	s.	d.
1851 . .	145,137	3	12	580,548	12	0
1852 . .	1,988,526	10	13	7,954,106	0	0
1853 . .	2,497,723	15	16	9,990,895	0	0
1854 . .	2,144,699	9	19	8,578,797	16	0
1855 . .	2,575,745	4	17	10,302,980	16	0
1856 . .	2,985,695	17	0	11,942,783	8	0
1857 . .	2,761,528	8	0	11,046,113	12	0
1858 . .	2,555,263	0	0	10,221,052	0	0
1859 . .	2,280,525	14	0	9,122,102	16	0
1860 . .	2,128,466	11	0	8,513,866	4	0
1861 . .	1,978,864	13	0	7,915,458	12	0
1862 . .	1,662,448	18	0	6,649,795	12	0
1863 . .	1,627,066	0	0	6,508,264	0	0
1864 . .	1,545,449	15	0	6,181,799	0	0
1865 . .	1,545,450	0	0	6,181,800	0	0
	30,422,591	0	5	£121,690,363	8	0

In addition to the above amounts, exported direct from the colony, 1,691,150 oz. of gold, produced in Victoria, are known to have been shipped through the Customs of New South Wales, Tasmania, and South Australia, besides a large amount (estimated at 2,250,000 oz.) that left the country in private hands.

The total exports of the colony, from the commencement of gold-mining operations to the end of 1865, will therefore be nearly as follows:—

	oz.	dwt.	gr.
Returned by Victoria Custom House	30,422,591	0	0
Not returned " " "	1,691,150	0	0
Taken by Private Hands	2,250,000	0	0
Total . .	34,363,741	0	0

The total value of the gold exported from the colony, up to the end of 1865, will consequently be nearly 138,000,000*l.*, exclusive of that absorbed by the currency, &c. of the country.[*]

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[*] The Colonial Secretary of Mines estimates the quantity of gold produced in the colony, from the first discovery of the gold fields, to the 31st December, 1865, at 30,998,071 oz.; value £123,992,284; but no allowance is made by him for the gold which had been sent off privately, or for the amount used and manufactured in the colony.

New South Wales.—Although the first practical discovery of gold was made in New South Wales, some weeks previous to its being found in Victoria, its total produce of the precious metal has been far less considerable than that of the sister colony. The general description of the gold fields of Victoria is equally applicable to those of New South Wales, but we are without any very definite information relating to the different workings in the latter colony, which are, however, much less numerous and extensive than those of Victoria. Among the principal mining districts of New South Wales may be mentioned Abercrombie, Summerhill, Ophir, Turon, Tamworth, &c. &c. The produce of gold from New South Wales up to the close of 1860, amounted to about 8,000,000*l.*, whilst Victoria had exported gold to the value of nearly 90,000,000*l.* during the same period. According to the Government returns, entitled "Statistical Abstract for the several Colonial and other Possessions of the United Kingdom, 1866," the export of coined and uncoined gold from this colony, up to 1864, had been nearly as follows:—

Period.	Quantity.	Value.
	oz.	£
1851	144,120	470,836
1852	818,751	2,660,945
1853	548,152	1,781,272
1854	237,910	773,209
1855	64,384	209,250
1856	46,999	156,151
1857	277,531	1,101,448
1858	443,462	1,773,851
1859	427,558	1,704,774
1860	472,886	1,878,588
1861	507,021	2,010,263
1862	741,055	2,984,269
1863	593,699	2,362,054
1864	740,048	2,952,471
	6,063,576	22,819,381

Although we have not been able to obtain the same amount of statistical information with reference to this colony, as is to be procured with regard to Victoria, it is probable that a very large proportion of the gold exported from New South Wales is the produce of other portions of the Australian Continent. The establishment of a mint at Sydney has naturally caused large quantities of gold to be imported, for the purpose of being converted into coin, but the total produce of the gold fields of New South Wales, since their discovery in 1851, is estimated at only 4,000,000 oz., and the present annual yield at about 320,000 oz.

South Australia and Tasmania also annually produce a certain amount of gold, but the quantity is comparatively small, since, from 1851 to the close of 1860, the total weight exported, including the exports from New Zealand, was represented by a money value of 374,000*l.*, of which New Zealand is supposed to have contributed more than one-half. The amounts and value of the gold exported from South Australia and Tasmania, including New Zealand, from the discovery of gold in those countries to the end of 1860, were nearly as follows:—

Period.	Quantity.	Value.
	oz.	£
1852	5,250	21,000
1853	5,250	21,000
1854	5,250	21,000
1855	5,250	21,000
1856	5,250	21,000
1857	15,639	62,556
1858	18,680	74,720
1859	16,431	65,724
1860	16,500	66,000
	93,500	374,000

Queensland is likewise, to a certain extent, a gold-producing country, and affords a small yearly yield of the precious metal, but we have been unable to procure any returns of its annual production. According to *Dicker's Mining Record* of 17th April, 1866, there is but one quartz-crushing establishment in that colony, and only one quartz-mining company now at work. This association holds the lease of three quartz reefs from the Crown.

The extent of the ground held is 400 yards in length, on the line of each reef, by a width of 200 yards. The Alexander Reef is twenty-two miles from Gladstone, on the water-shed of the Boyne River. A crushing had been made of 400 tons, which paid satisfactorily. There was only one very indifferent mill of 10 horse-power in the country. The charge for crushing was 25s. per ton. On this reef there were five shafts, varying in depth from 80 to 100 feet; the thickness of the vein varies from two to four feet. The reef forming the second lease is on Bell's Run, twelve miles from Gladstone, on the water-shed of the Caliope. There had been six shafts sunk, varying in depth from 50 to 70 feet, and the thickness of the lode is from three to eight feet. The proprietors had sent two tons to Sydney to be crushed, but the result had not been announced at the date of the despatch leaving the colony. The third lease is on a quartz reef, on the range dividing the waters of the Boyne from the Caliope. Not much, however, had been done here up to the spring of 1866, although two shafts were then in course of sinking.

NEW ZEALAND.

The first authentic discovery of gold in New Zealand was made at Massacre Bay in 1842, by an exploring party under Captain Wakefield, but did not at the time attract much attention. No further discoveries of this metal were announced until 1852, but in that year gold was almost simultaneously found in the Provinces of Auckland and Otago, and, from the important results which had then been obtained from the Australian gold fields, excited a considerable amount of public attention. The Auckland discovery was made at Coromandel, but only about 1,100 ounces of gold were obtained, and the district was shortly afterwards abandoned. In 1856, attention was called by the Surveyor-General of New Zealand to the existence of gold in the sands and gravels of the Mataura River, and in the same year discoveries were made at Motaeka, in the Province of Nelson; whilst inthe following year, the gold field at Aorore, Massacre Bay, again came into notice. A rush of the population to the gold regions was the result of these discoveries, and about a thousand persons were for some time employed in the diggings with varying success; but the severity of the season, and want of practicable roads, soon led to the discouragement of a large number of the miners. The exports from this district, up to the end of 1858, amounted to 16,473 ounces of gold. The richest diggings on the Aorore gold field were those on the Slate River, which takes its rise in the Anatoki Range, and afterwards falls into the Aorore. This river has high and precipitous banks, composed of granite, slate, and quartz; the gold being found, associated with osmiridium, in a yellow sand met with in the bed of the stream.

In the latter part of 1857, the existence of gold in Otago was made known; and the Sub-Assistant Surveyor, Mr. Gillies, and party, found gold in a creek running between the Waikioi and Makerewa Bush, and emptying itself into the Makerewa. About the same period, Mr. Garvie, another Sub-Assistant Surveyor, found traces of gold in many of the sands and gravels of the south-eastern district of the Province. In March 1858, Mr. Garvie brought into Dunedin the first important specimens, which were obtained by Mr. Buchanan from the river beach of the Dunstan gorge, which four years later proved so highly remunerative to two old Californian miners, named Hartley and Reilly, and which ultimately led to the development of the most extensive gold field in the Province. In the same year, gold was discovered in the surface gravel near the mouth of the Tuapika River, and also in the River Lindis.

Further discoveries continued to be made in the Province of Nelson, and in 1859 some nuggets, weighing from two to nine ounces, were found in the Rocky River.

Gold was found in considerable quantities in the River Lindis in March 1861, in the form of water-worn nuggets of the size of beans as also, in a finer state of division, in the Kakanui, near Moeraki. In June of the same year, a discovery was made from which may be dated the importance of Otago as a gold-producing country. Mr. Gabriel Read was led by curiosity to attempt the verification of the reported presence of gold in that district, and in the course of his expedition examined the ravines and tributaries of the Waitahuna and Tuapika rivers. With a tin dish and a knife for his only tools, he collected seven ounces of gold in about ten hours, and ascertained the existence of the precious metal in many of the creeks and gullies. These discoveries of Mr. Read were shortly afterwards confirmed by the results obtained by other explorers, and caused a great rush to the district, where discoveries followed each other in rapid succession, fully establishing the fact of the existence of a valuable and extensive gold field.

Early in 1862, fresh discoveries were made at Coromandel, where a large number of miners congregated, and several companies were organised for working quartz veins. The gold diggings on the western coast of Nelson also gave promise of being highly productive, and some very large prospects were obtained. In the month of August a discovery was made in the Province of Otago which exceeded in importance those of the previous year. In February, two men, who had been miners in California, started on a prospecting tour up the Molyneux River, and found gold so readily, that, to use their own words, "they had nothing to do but to set the cradle on the edge of the river and keep it going from morning till night, as one man could get rich wash dirt to feed the cradle as fast as the other could wash it." These men, Hartley and Reilly, as the result of three months' work, brought into Dunedin 87 lbs. of gold, and received from the Provincial Government a bonus of 2,000*l.* for making known the locality from which it had been obtained.

In the latter part of 1862 and the beginning of 1863, the area of the gold field of Otago was much extended, and in the latter year large amounts of gold were obtained, and immense areas of auriferous ground opened up, on the Wakatipu Lake and its tributaries. Towards the latter part of 1863, further and important discoveries were made on the western coast of Nelson, and in the early part of the following year the Matakitaki Diggings, in the same province, came into considerable notice.

In April 1864, the first discovery of gold was made in the Province of Marlborough, in the River Wakamarina and its vicinity; but disastrous floods, and the impenetrable nature of the country, have prevented the rapid development of this gold field. During the same year, gold was obtained from the banks of the Teremakau River to the west of Canterbury, and various other localities along the coast.

In the year 1865, another very important discovery of gold was made in the Hokitika River, on the west coast of Canterbury, the yield from which, up to the present time, has been very considerable; and as further discoveries are being made in the district, it is probable that this river, and the district around it, will turn out to be a rich and important gold field. Gold mining is now being successfully carried on at Coromandel in the Province of Auckland, at Massacre

Bay, and on the Buller, Lyell, Wangapeka, and other western streams in the Province of Nelson; in the Grey, Teremakau, and Hokitika rivers, and over a large extent of the western coast of the Province of Canterbury, as well as over a vast area in the Province of Otago.

In New Zealand, as in all other gold-bearing countries, the zones of clay slate and mica slate appear to be the original sources from which the supply of the precious metal has been chiefly derived, but the largest portion of that at present obtained has been procured from workings in the alluviums, some of which, like those of California and Australia, extend under a capping of volcanic rock.

The geological age of the auriferous drifts of New Zealand is probably that of the gold-bearing alluviums of Australia, although this does not appear to have been as yet distinctly proved.

Quartz mining is as yet in its infancy in the colony, but several promising reefs have been opened, and are expected to afford satisfactory returns. The most important quartz mines at present known are at Waipori. A large proportion of the gold produced in New Zealand is obtained by sluicing, and so favourable have been the results of this class of mining, that it may be fairly anticipated, that the Province of Otago will be a large gold-producing country for many years to come. River mining is profitably followed in various parts of the country, particularly in the Wakatipu district, and in the beds of the Molyneux and neighbouring streams. As these deposits gradually become exhausted, the attention of miners will, doubtless, be more exclusively directed to the thicker formations of auriferous drift.

The following statistics relative to the gold production have been compiled from official sources:*—

TABLE I.

AMOUNT OF GOLD EXPORTED.

	oz.	dwt.	gr.
1861-1862, 1st August to 31st July	457,239	10	6
1862-1863 " "	514,385	17	0
1863-1864 " "	497,031	9	0
1864-1865 " 31st March	231,010	11	0
Total quantity exported from Otago . .	1,699,667	7	6

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* New Zealand Exhibition, 1865, Reports and Awards of Jurors.

In addition to the above, 63,970 oz. of gold, the produce of the gold fields of Otago, were exported from other ports in the colony, making the grand total exported 1,763,637 oz. 7 dwt. 6 gr., value 7,054,544*l.*

TABLE II.

Showing the quantity and value of gold from the gold fields in each Island, and the whole of New Zealand, exported from the Colony, from 1st April 1857 *to 31st December* 1864.*

Produce of.	During 1864.		From 1st April 1857 to 31st December 1863.		Total exported from New Zealand, to 31st Dec. 1864.	
	oz.	£	oz.	£	oz.	£
North Island	3,448	10,552	6,076	19,323	9,524	29,875
South Island	476,723	1,847,295	1,263,112	4,894,560	1,739,835	6,741,855
Total from New Zealand	480,171	1,857,847	1,269,188	4,913,883	1,749,359	6,771,730

The following table gives the approximate yields, in lbs. troy, of the principal gold-producing countries at the commencement of the present century, and for the years 1850, 1860, and 1865. In cases where the returns for the year indicated could not be obtained, the produce for the nearest years, for which they could be procured, has been substituted. The quoted produce of the mines of the United States and of the British Possessions, may, however, be regarded in each instance as being nearly correct, except that the return for California and the neighbouring States and Territories for 1865, is probably somewhat under the truth, since it is exceedingly difficult to ascertain the precise yields of Idaho, Montana, Colorado, and some other outlying districts. After each absolute sum is given its relative weight, in comparison with the grand total produced throughout the world:—

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* The returns state: "The value has been calculated on the uniform estimated rate of 3*l.* 17*s.* 6*d.* per oz., with the exception of the gold from the North Island, for which the ascertained value has been allowed. This table has been compiled from the *Monthly Returns of Gold Exported*, which do not in all cases exactly correspond with the Quarterly Trade Returns of Exports from the Colony, as furnished by the collectors of their respective ports, from which Table I. was constructed."

TABLE SHOWING APPROXIMATE PRODUCTION OF THE PRINCIPAL GOLD FIELDS OF THE WORLD.

	1800.		1850.		1860.		1865.	
	lbs. troy.	ratio per cent.	lbs. troy.	ratio per cent.	lbs. troy.	ratio per cent.	lbs. troy.	ratio per cent.
Russian Empire . .	1,440	2·7	65,600	19·0	66,000	11·3	69,500	12·4
Austrian Empire and	3,500	6·5	5,600	1·6	5,500	1·0	5,500	1·0
Rest of Europe			100	...	350	...	375	
Southern Asia . . .	10,000	18·5	25,000	7·3	25 000	4·3	25,000	4·5
Africa.	600	1·2	4,000	1·1	4,000	0·7	4,000	0·7
Chili	7,500	13·8	34,000 *	9·9	34,000	5·9	34,000	6·1
Bolivia	1,600	3·0						
Peru	2,400	4·4						
New Granada . . .	12,600	23·4						
Brazil	10,000	18·5						
Mexico	4,300	8·0						
California & neighbouring States & Territories . .	...	...	208,000	60·2	187,000	31·9	210,000	37·5
Rest of United States	...	...	2,950	0·9	1,020	0·2	140	
Nova Scotia . . .	...	...	...	...	...	...	2,072	0·4
British Columbia. .	...	...	...	...	20,000	3·4	11,600	2·1
Australia	...	...	...	...	217,500	37·0	156,000	27·9
New Zealand . . .	...	...	...	...	25,000	4·3	41,400	7·4
	53,940	100	345,250	100	585,370	100	559,567	100

* The yields of the different members of this group vary considerably from year to year, but the aggregate produce is believed to remain tolerably constant.

CHAPTER VIII.

GOLD WASHING IN CALIFORNIA AND AUSTRALIA.

PLACER MINING—THE PAN—ROCKER—LONG TOM—PUDDLING BOX—SLUICE—RIVER MINING—BEACH MINING—WATER SUPPLY—EUREKA CANAL—HYDRAULIC MINING—AMOUNT OF WATER REQUIRED—COST OF WATER, AND METHOD OF MEASUREMENT—DRY WASHING.

GOLD mines may be divided into two distinct classes, viz., Placer Mines, in which the metal is found, in a more or less water-worn condition, embedded in earth, clay, sand, or gravel; and Quartz, or Vein Mines, in which gold is met with disseminated in its original gangue, or matrix.

In the former, the gold-producing material is called "pay dirt," which, on being subjected to the action of water, becomes disintegrated, and the lighter portions are mechanically carried off, whilst the gold, from its greater specific gravity, remains behind. In the latter, on the contrary, after first obtaining the rock by the ordinary operations of mining, as practised with regard to other metals, it has to be reduced by mechanical means to the state of a finely-divided powder, before the associated gold can be collected, either by washing or amalgamation.

The amount of skill and capital necessary for the successful prosecution of placer mining, is usually less than is requisite for carrying on quartz mining, on a remunerative scale; and as placer mines are generally those to which attention is first directed in a new country, and from which remunerative returns are most readily obtained, we shall, in the first place, describe the processes employed for extracting gold from the various alluvial deposits, and subsequently pass on to the consideration of quartz mining, and the extraction of the precious metal from the veinstone constituting its original matrix.

Water is the great agent by the aid of which placer mining is carried on: with a large supply, the operations of the miner can be cheaply and rapidly conducted; but without water, or with only a limited amount, a claim that would otherwise have been highly productive, may either become valueless, or only capable of affording very irregular returns.

Placer mines are of two distinct classes, the shallow and the deep; shallow, or surface diggings, are generally found in the beds of ravines or gullies, on the bars and in the beds of modern rivers, and on shallow flats. In the latter, the pay dirt is often found at great depths from the surface, and is not unfrequently covered by thick beds of lava, or volcanic ash, as in the case of the deposits under Table Mountain, Tuolumne County, and near Nevada City, California.

In the deeper placers, the auriferous drifts are reached either by shafts of considerable depth, or by means of levels, or tunnels, driven in from some neighbouring valley. The pay dirt, after being thus extracted, is conveyed to the surface, in order that it may undergo the usual process of washing. In other instances, hydraulic mining is resorted to, in which case jets of water, under great pressure, obtained from a high column, are directed against the deposits of sand and gravel, which are thus not only disintegrated, but finally carried away by the current. This is the most economical and expeditious method of working placer mines, when a sufficient supply and pressure of water can be obtained, and there is also enough declivity below the auriferous beds to allow of the resulting detritus being readily disposed of. The pay dirt is almost invariably covered by various layers of barren clay and sand, which are, in the shallow diggings, removed by the use of the pick and shovel; but, in hydraulic workings, the whole is washed away by the force of a stream of water playing against it, and any particles of gold which it may contain are caught in the sluices through which the lighter materials pass. In many of the deep placers, after reaching the pay dirt by means of shafts or tunnels, it is extracted by a system of levels and headings, not unlike those employed for working a coal seam in this country.

In California, besides classifying the placer mines as shallow and deep, they are again subdivided into hill, bench, flat, bar, gulch, and river diggings, with reference to their topographical position as regards the surrounding country. Hill diggings are in the sides of hills; bench diggings are on narrow benches on the declivities of hills, and above the level of existing rivers; flat diggings are situated on flats or small plains; bar diggings are usually in collections of sand and gravel, on the sides of streams, and, under ordinary circumstances, above the surface of the water; gulch claims are found in ravines and gullies, through which no water passes, except in times of extraordinary floods; river diggings occur in the bottoms of rivers, and can only be worked after diverting the water from its original channel.

Placer claims are likewise spoken of as sluice, hydraulic, and tunnel claims, dry diggings, &c., in accordance with the means employed for reaching the auriferous deposits, and the methods adopted for the separation and collection of gold. In the early days of Californian and Australian gold mining, when paying diggings could generally be found near the surface, and before the large deposits of pay dirt lying at great depths below the ground had been discovered, the greater portion of the gold produced was obtained from shallow workings. These have now, to a great extent, become exhausted, and in Australia shafts are sunk to the older deposits, whilst in California enormous aqueducts have been constructed, and hydraulic mining is extensively introduced. In both countries, therefore, at the present time, the greater portion of the gold obtained is the result of deep mining, the shallow placers being generally abandoned to the Chinamen, who are satisfied to wash over the dirt which has been already passed through the machines, or has escaped the attention of the ordinary miner.

The object of the miner is generally to obtain the largest quantity of gold in the shortest possible time, and with the least amount of labour and expense, rather than to extract the total amount contained in the material on which he operates; and, consequently, without taking into consideration the necessary loss incident to the imperfect nature of the appliances employed, his refuse will naturally retain a certain proportion of the gold originally present. The falling abroad, however, of lumps of clay, by the action of the weather, constantly exposes fresh particles of the precious metal, and from this cause the residues frequently afford an amount of gold that could not have been otherwise obtained from them. Many contrivances have been introduced with a view to obtaining the whole, or a large proportion, of the gold present in the pay dirt, but, in the majority of instances, the result has been a failure; and if an increase of yield has sometimes been obtained, the additional gold extracted has frequently been more than compensated for by the extra expense entailed.

Whilst the supply of shallow auriferous dirt was sufficient to furnish employment for the whole labouring population, the gold produced was almost exclusively obtained either by solitary diggers, or by small private companies of working men, whose capital consisted rather in their united strength, than in an accumulation of money. Things went on in this way until the richer and more easily-worked alluviums had become partially exhausted; when it was found that to carry on successfully the deeper diggings, combinations on a somewhat more extensive scale became necessary, and that capital and intelligence, as well as strength and endurance, were required by the gold miner. While there was no difficulty in finding shallow claims that would pay from 25*s.* to 30*s.* per day to the hand, it was almost impossible to obtain the labour necessary to carry on either deep digging or quartz mining with any chance of success; but when these began to be worked out, this difficulty gradually disappeared, and attention became directed to operations on a more extensive scale, generally conducted either at the expense of a co-operative company, or joint-stock association. In this way, the usages and exigencies of a newly-discovered gold region rapidly change and become more assimilated to those of more commercial and longer established communities. A lucky miner becomes a capitalist, and in the gold regions, as in all other places, money commands labour, and large operations are commenced, often to the doubtful advantage of the speculators, but to the manifest benefit of the general community.

Having thus briefly alluded to the general conditions of recently discovered gold regions, we will proceed to describe the various appliances employed for the separation of gold from the different earthy materials with which it is associated, beginning by the cheaper and more simple, and afterwards treating of the more complicated and efficient arrangements now generally employed.

The Pan.—This is the indispensable companion of the gold miner, and is used by him in all branches of his business, either for washing, or as a receptacle for gold, amalgam, or rich dirt. It is made either of stiff tin plate, or of thin sheet iron, and is at bottom about fourteen inches in diameter. A sheet iron pan is generally to be preferred to one made of tin plate, not only because it is stronger, but also because it is unattacked by mercury, if brought in contact with that metal. The top is from three to four inches wider than the bottom, and consequently the sides, which are about five inches in depth, and strengthened at the edge by a thick wire, have a considerable inclination outwards. In order to wash with the pan, it is first about two-thirds filled with dirt, and then placed in a water-hole, which should not be more than about a foot in depth, in order that the vessel may rest on the bottom, whilst the miner stirs up its contents with his hands. If the water be too deep for this, the pan may be held in one hand, and stirred with the other, but it is more convenient, and less tiresome, to be enabled to let it rest on the bottom. The dirt having been placed in the pan, and the pan itself filled with water, the miner inserts his hands into it, and raises and stirs the dirt, so as to make every part of it thoroughly wet. Having done this, he now takes the pan in his hands, holding it by the two opposite sides, but somewhat behind its greatest diameter; and raising slightly the part which is towards him, and consequently depressing the outer edge, he commences to shake it from side to side in such a way that all the dirt is beneath the surface of the water, and at the same time, by a sort of circular motion, he allows a small portion of it to pass over the edge at each oscillation. The earthy particles, together with the fine sand, soon become suspended in water, and gradually pass over the side in the form of thin mud, whilst the gold, the tough clay, and large stones remain behind.

The stones collect on the top of the sand and clay, and are scraped off with the fingers, while the lumps of clay are rubbed between the hands, and thus prepared for being taken up by water during the next washing. The process is thus continued, the pan being gradually raised in the water, and its outer edge further depressed, until the greater portion of the clay and stones have been removed, and the gold remains, together with a little earthy matter and black sand. At this stage of the operation, the pan is nearly filled with water, and after being removed from the pool in which the washing has been hitherto conducted, is shaken, so as to get the last portions of clayey matter in suspension, and the water carefully poured off, as shown in Fig. 7.

The outer edge of the pan only is now immersed in water, and a further portion of the clayey matter removed by careful washing, leaving the gold, and a small quantity of heavy titaniferous iron sand, in the angle formed between the bottom and side. This black sand, which is mixed with the finer particles of gold, is so heavy that it cannot be separated from the metal by washing, but can, to a great extent, be removed by a simple process of blowing. In order to do this, it is allowed to dry, and a small quantity of it placed in a *blower*, which is a sort of dish, or shallow tin scoop, open at one end. The miner now holds the blower with the open end from him, and with his mouth directs a gentle stream of air along the surface of the mixture of gold and fine sand, taking care so to regulate its force as to remove the latter without causing a loss to the former. To effect this, the blower must be gently shaken, so as to change the position of the particles, and bring them all, in turn, under the influence of

FIG. 7.

WASHING WITH PAN.

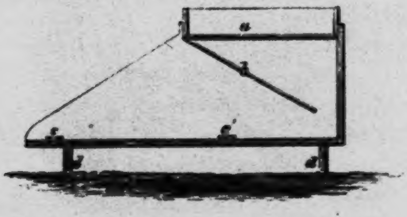
the current of air; and although the gold cannot be thoroughly cleansed in this way, any remaining particles are readily removed by a magnet.

The pan is always employed for washing up the rich dirt which collects in the cradle and sluice heads, and is exceedingly convenient for cleaning small quantities of gold-bearing sand collected in the different arrangements made use of in placer mining. Amalgam can be separated from sand by washing, almost as readily as gold, and the pan is therefore frequently used for this purpose. Panning is apparently a very simple process; but in order to use a pan so as to wash quickly, and at the same time without loss of gold, considerable practice is required.

The Cradle.—This apparatus somewhat resembles in size and shape a child's ordinary wooden cradle, and stands on similar rockers. The box is usually about forty inches in length and twenty in width, with one end from fifteen inches to two feet in height, the sides being sloped off at the lower extremity like those of a coal-scuttle. The construction of a cradle and the method of using it will be understood by reference to Figs. 8 and 9.

At the upper end of the cradle is the hopper, or riddle box, *a*, twenty inches square and six inches in depth, of which the bottom is composed of sheet iron, thickly perforated with holes half an inch in diameter. This is not fastened to the cradle, but can be lifted on and off at pleasure, and fits into it so as to be quite steady when in its right position. Beneath the riddle is placed an apron, *b*, made by stretching a piece of canvas on a framework resting on fillets, inclined from the bottom edge of the riddle towards the head of the cradle, whilst across the bottom are nailed two riffle bars, *c*, about three-quarters of an inch in height, one towards the middle, and the other at the lower end.

FIG. 8.



SECTION OF CRADLE.

FIG. 9.

WASHING WITH CRADLE.

In order to work with this apparatus, which stands on rockers, *d*, the dirt is shovelled into the hopper; the cradler sits or kneels by the side of his machine, and whilst, with a ladle or dipper in one hand, he pours water upon the dirt, he with the other gives a rocking motion to the cradle. By the action of the water and motion together, the dirt is rapidly disintegrated, and passing through the riddle falls upon the apron, by which it is carried to the head of the cradle box, and from thence flows along the bottom, and finally escapes at the lower end, leaving the gold, black sand, and heavier particles of gravel, behind the riffle bars nailed across it. The pay dirt usually contains a great many large stones, which remain on the hopper, from whence those which are so large as to give an unpleasant jerk to the cradle in rocking are removed by hand. The smaller ones, however, are allowed to remain until a hopperful of clean stones has accumulated, when the cradler, rising from his seat, first looks them over to see that there is no gold among them, and then takes out the riddle box, and with a jerk throws out its contents. The rocking motion of the cradle not only assists the disintegration of the dirt, but also tends to keep the sandy deposit behind the riffle bars from becoming too solidly packed, which, by preventing the particles of gold from settling at the bottom, would cause a considerable loss.

The operation of washing with the cradle is merely a repetition of the processes above described; but, in order to prevent loss of gold, it is necessary to clean up the cradle box from two to three times a day, according to the richness and nature of the wash dirt which is being worked. To do this, the hopper is first lifted off, and the apron removed, in order to get readily at the bottom of the cradle, which is then carefully scraped with an iron spoon, and the dirt collected in a pan for the purpose of being subsequently panned out. The larger proportion of the gold naturally collects above the riffle bars; and when it is in a very finely divided state, it is sometimes found advantageous to place the hopper over the lower end of the cradle, which will thus take double the length of apron, and oblige the stuff washed to pass over a longer surface before leaving the machine. If, in this case, the apron be made of a thick woollen cloth, a considerable amount of fine gold will be retained on its surface, and on cleaning up the machine, it can be readily washed off into a pan. For the convenient supply of a cradle, water should be conducted, by means of a small ditch or gutter, to a pit sunk near its head, serving as a reservoir, and from which it is dipped out by means of a basin-shaped ladle, provided with a short wooden handle, and capable of containing from three to four quarts.

The difference of level between the upper and lower ends of the machine should, under ordinary circumstances, be about two and a half inches, but this may be slightly varied in accordance with the fineness of the gold, and the nature of the dirt to be washed. The amount of dirt that can be washed by one man in a day, will evidently depend on the quantity of clayey matter it contains, and varies from one to three cubic yards. The dirt is shovelled into a pan or bucket, and from thence thrown into the hopper, the miner estimating the amount of work done by the number of pans or buckets washed.

Although the cradle is frequently worked by one man, washing by this machine can be more expeditiously and cheaply conducted by two persons, since, in order to keep it in constant operation, there is always sufficient work for a cradler and a shoveller. In that case, one of the miners attends the cradle, whilst the other, who is provided with a couple of buckets or pans, fills them alternately with dirt, always keeping a full one near the cradle, so that he can at once pick it up and empty it into the riddle box. When a cradle is worked only by one man, he has continually to stop working in order to discharge the stones from the riddle, and fetch more dirt, and during this time the sand and clay retained by the riffle bars are liable to pack, and have to be loosened with a spoon, or scraped, before again beginning to wash. It has been found in practice that the weight of water required for working a cradle is at least three times that of the dirt washed, and consequently, where there is no means of conducting a supply to the cradle, it is taken to water, and the wash dirt transported to it. It is of great advantage to the miner when he can place his cradle within a few feet of the pit from which he obtains his pay dirt, and it is also essential that he should have an adequate supply of water and a sufficient fall to enable him to get easily rid of his tailings.

The rocker is neither an expeditious nor economical method of washing, as it not only loses fine gold, but also gets through but one-fifth the amount of work that the same number of hands can perform with a tom, and less than one-tenth of that which an ordinary sluice will accomplish, but is nevertheless peculiarly adapted to certain descriptions of diggings. In all gold-producing countries there are numerous little gullies and ravines containing coarse gold, but in which no water can be obtained for washing, except immediately after heavy rains, and then only for a few days at a time; and in such situations the cradle can often be used with advantage. It is in the first place cheap, does not require a large quantity of water, and being exceedingly portable, can, when the supply fails, be readily removed to the bank of some pool, where a supply of pay dirt can be obtained.

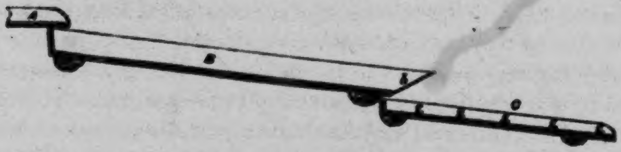
The great defect of the cradle is the tendency of the sand to pack behind the riffle bars, which causes the bottom of the box to assume a plain even surface, over which gold is liable to pass off and become lost. It is consequently necessary, after the machine has been allowed to stand, if only for a few minutes, without rocking, to stir up the sand in the bottom before recommencing the operation. Mercury is sometimes employed in the rocker, but this is not general, and its use in this apparatus is not generally to be recommended. In the early days of Californian and Australian gold mining the rocker was very commonly employed; but now that the supply of rich shallow pay dirt has become nearly exhausted, it is but little used by European and American miners, although still a general favourite with Chinese diggers.

The Tom.—The tom or long tom was almost universally employed in the Californian placers up to the year 1852, but is now rarely met with. It consists of a roughly-made wooden trough, or spout, about twelve feet in length, twenty inches in width at its upper end, and gradually widening to thirty inches at the other extremity, with its bottom covered by a plate of iron to prevent wearing. The sides, from eight to ten inches in depth, are cut off obliquely from the bottom upwards, so that the wide end may be closed by an inclined riddle of punched sheet iron, precisely similar to that forming the bottom of the hopper of an ordinary rocker. This trough is supported on trestles or logs, so as to have an inclination from the head downwards of about twelve inches, and beneath the strainer of sheet iron is placed the riffle box, which may be used either with or without quicksilver, and from which the rich dirt is, from time to time, cleaned out and panned up, as in the case of the rocker.

The general arrangement of the tom and its riffle box will be understood by the aid of Figs. 10 and 11, which show the method of fixing this apparatus.

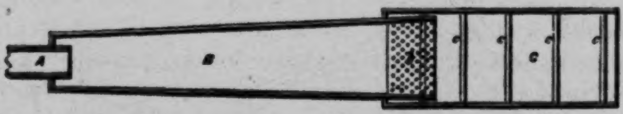
A stream of water is brought by the spout, A, on to the tom, B, and the dirt is thrown in near its head by one man, whilst another keeps it constantly stirred, either by a shovel, or by a fork provided with numerous prongs, with which he from time to time removes the large stones, and throws back against the current such pieces of clay as continue to hang firmly together. The number of men working at a time varies from two to four, according to the tenacity of the dirt, and the quantity of water available. The small stones which gradually accumulate in the angle between the bottom and the perforated iron plate, *b*, are removed when necessary, and a fresh supply of pay dirt is continuously shovelled in at the head of the trough.

FIG. 10.



SECTION OF TOM AND RIFFLE BOX.

FIG. 11.



PLAN OF TOM AND RIFFLE BOX.

This arrangement is most applicable to diggings yielding but a limited amount of pay dirt containing coarse gold; but even when quicksilver is used behind the riffles *c*, in the riffle box C, the loss of fine gold is considerable, although the constant falling of the water on it generally prevents packing.

The sluice, which has now generally superseded the tom, is not only capable of washing a much greater quantity of dirt in a given time, but is also attended with a less loss of gold.

Puddling Box.—The puddling box is a rough wooden case, usually about six feet square, and eighteen inches in depth, which is used for disintegrating very tough clay. The dirt is thrown into this box, and a considerable quantity of water added, with which it is stirred up by a rake, having very long and strong teeth, until the whole of the clay is held in suspension, when a plug, a few inches from the bottom, is removed, and the slimy matters run off. More clay and water are now introduced, and the operation is repeated until the box becomes filled with gravel and coarse sand to the level of the plug hole, when it is removed for the purpose of being washed up by the pan, cradle, or some similar appliance. In California, the puddling tub is only employed in diggings carried on on a very limited scale, and never where the sluice or the hydraulic process can be introduced. In Australia, on the contrary, where the supply of water is often scanty, washing by the aid of a puddling machine is generally resorted to. For small operations, half a porter barrel is sometimes used, the clay being stirred with a shovel; but in works conducted on a more extensive scale, an arrangement not unlike a brickmaker's pug-tub, set in motion either by horse or steam power, is generally used. This machine, when worked by steam power, consists of a large shallow tub with an upright shaft standing in its centre, provided with strong rake-like arms, set in motion by a mitre wheel attached to the perpendicular shaft. From these tubs the thin mud is tapped off, in the same way as from the puddling box, and the residues afterwards subjected to some process of washing. These machines are so general in Australia that in 1860 no less than 3,958 of them, worked by horse power, were in use in Victoria alone.

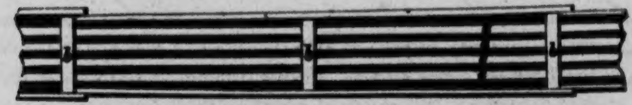
The Sluice.—This arrangement is now almost universally employed by Californian miners, and by it are, probably, collected at least eight-tenths of the gold furnished by the placer mines of that country. The sluice is generally a long wooden trough, having a considerable inclination or declivity, into which the pay dirt is shovelled, and through which a rapid stream of water is continually flowing. The bottom of this trough is provided with a series of riffles, generally containing mercury, by which the gold is retained, whilst the clay, sand, and gravel are carried off by the force of the current. The ordinary sluice is composed of a series of rough wooden boxes, each twelve feet in length, generally varying from sixteen to twenty inches in width, and from ten inches to a foot in depth, made of inch-and-half pine planks. In order to facilitate the making of these sluice boxes, the boards for the bottom are sawn, at the mill, four inches wider at one end than at the other. This enables the narrow end of one box to be fitted into the wide end of another, and in this way a sluice, several hundred feet in length, can be rapidly put together, or, if necessary, taken apart and removed. The descent of a sluice is called its *grade*, and is commonly from ten to eighteen inches on each box of twelve feet in length, and, consequently, a sluice having the latter inclination would be said to have an eighteen-inch grade. The grade to be employed is regulated by the position and length of the apparatus, and the nature of the dirt to be washed. The greater the descent, the more rapid will be the current, and, as a consequence, the greater is the danger of fine particles of gold being carried off by the water and lost. If the angle at which the boxes are set be considerable, the dirt will be rapidly disintegrated, but the danger of losing fine particles will be increased; whilst if the stuff to be washed contains much clay, a small inclination will not easily effect its disintegration. Generally speaking, therefore, a sluice with a rapid current of water requires to be made longer than one set more nearly on a level, and, for this reason, where a long sluice cannot be employed the inclination must be diminished.

Economy and the convenience of working render it necessary that the sluice should not be raised too high above the surface of the ground, and therefore the angle at which it is laid is somewhat influenced by the conformation of the country. There are, obviously, various considerations to be attended to in finding the proper grade; but in most instances a fall of less than ten inches, or more than twenty inches, on the length of a twelve-foot box, is not regarded as suitable for the ordinary sluice. In many claims the pay dirt contains large blocks of stone and boulders, each weighing from fifty to several hundred pounds, which have to be passed through the sluice, either whole or after being broken, and in such cases a large body of water and a rapid current are essential. The upper part of a sluice is sometimes made steep in order to effect the disintegration of the dirt, whilst the lower is placed at a less inclination for the purpose of collecting the gold, and this arrangement is often found advantageous. The ordinary clay met with in pay dirt is entirely taken up in suspension by the water of a sluice with a moderate grade, within the first two hundred feet, and the remainder of the boxes beyond that point are only useful in collecting gold. It is scarcely necessary to observe that when the gold is coarse, the inclination may be safely made greater than when it occurs in a finer state of division. In some claims the clay is so extremely tenacious that it will roll into balls, which are carried the whole length of an ordinary sluice without being much diminished in size. This has to be carefully avoided, by breaking up the lumps at the head of the sluice, since balls of plastic clay passing through the boxes not only do not give up the particles of gold they may contain, but are also liable to pick up others over which they may pass in their course.

Sluice boxes are always provided with some sort of false bottom for the purpose of retaining the gold, which would otherwise not only be taken away by the force of the current, but the bottoms themselves be rapidly worn out by the attrition of the stones and gravel passing over them.

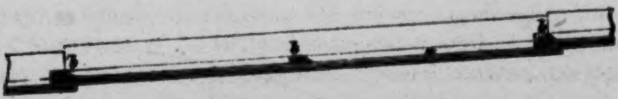
In the majority of cases the false bottoms employed are composed of longitudinal bars *a*, from two to four inches in thickness, from three to seven inches in width, and about five feet six inches in length. These are wedged in the boxes, from an inch to two inches apart, by the cross pieces *b*, so that two lengths of bars are fitted in the bottom of each box, as seen in Figs. 12 and 13.

FIG. 12.



PLAN OF SLUICE BOX.

FIG. 13.



SECTION OF SLUICE BOX.

The whole of the boxes are thus fitted with these bars, and the bottom of the sluice consequently represents a series of rectangular depressions, of the thickness of the bars, and several feet in length, in which the gold, mercury, and amalgam are caught. It is evident that the larger pieces of gold will be readily caught by such an arrangement without the use of quicksilver; but in order to retain the finer particles, the employment of this substance becomes essential.

When the sluice boxes have been all joined together, and the bars wedged into the bottom of each, the apparatus is ready for working; for although the boxes are generally made of planks rough from the saw, the swelling of the wood, on the introduction of water, and the clay which gets into the joints, as soon as the wash dirt is thrown in, quickly closes them, and renders the whole arrangement water-tight. The pay dirt is now shovelled in at the head, the number of men employed being regulated by the size of the sluice, and the nature of the dirt, which the current rapidly disintegrates, carrying off the clay in suspension, and rolling the pebbles and boulders onward by the force of the stream. The first dirt thrown in closes the joints of the troughs, and fills the spaces between the riffles, but nevertheless leaves a sufficient number of pits and inequalities for retaining the particles of gold and amalgam.

The amount of dirt which can be thrown into an ordinary sluice in a day by one man, depends on the compactness and state of aggregation of the deposit, and varies from about two to five cubic yards. About one hour and a half or two hours after the commencement of sluicing, some mercury is poured into the head of the apparatus, from whence it gradually finds its way downwards, in the direction of the current, but is still chiefly retained by the upper boxes of the series. Smaller quantities of quicksilver are also introduced between the bars, along in various places in the boxes; and the greater the amount of fine gold present, the larger must be the quantity of mercury used.

When the gold contained in the dirt is exceedingly fine, an amalgamated copper plate is sometimes resorted to. This plate is generally three feet in width, and six in length; is set nearly level, and, when the sluice is a very large one, the stream is frequently divided into two or three separate portions, each of which is conducted over a distinct amalgamated plate.* A well-amalgamated copper plate is considered as effective for saving fine gold, as an equal surface of pure mercury, and is not only cheaper, but also more easily managed. The copper plate is, in most instances, placed at a considerable distance from the head of the sluice, and the dirt and water falling upon it first passes through a sheet iron screen, having apertures half an inch in length, and a sixteenth of an inch in width. The amalgamation of the copper plate is effected by first washing over its upper face with dilute nitric acid, and then rubbing on, with a rag, quicksilver on which a little diluted nitric acid has been first poured, so as to form a certain amount of nitrate of mercury. When a plate has been thus well covered, this operation need never be repeated, it being only necessary to sprinkle its surface occasionally with a little fresh quicksilver, in proportion as the gold caught converts it into a solid amalgam.

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* For the purposes of hydraulic mining sluices from four to seven feet in width are frequently employed.

In order that these plates should act satisfactorily, it is essential that the current should be slow, and the water shallow, since otherwise a considerable portion of the fine gold might escape without coming in contact with the face of the plate; and it is for this reason that in large sluices it is usual to divide the stream, and to pass each portion over a separate surface of amalgamated copper. Wherever a particle of gold has attached itself to the face of an amalgamated plate, others will be found to arrange themselves around it, evidently becoming more readily attached to those portions of the surface on which gold has been already caught, than on those on which no deposit of this metal has previously taken place. When a newly amalgamated plate is first used, its surface is apt to become tarnished by the formation of subsalts of copper, which forming a green slime, interfere with the adhesion of gold. This should be carefully scraped off, and the place from whence it has been removed rubbed with a little fresh mercury. The larger the amount of gold deposited on the surface of a copper plate, the better it is considered for the purpose of arresting the progress of fine particles of that metal; but as the accumulation of a very large quantity of auriferous amalgam might give rise to losses through theft, it is injudicious to allow too great a thickness to accumulate before being removed. For this purpose, the plate is taken up and heated over a fire, until the hand cannot bear to remain on it beyond about a second, when the amalgam becomes softened and loosened, and can be easily removed by scraping. The plate, after cooling, may now be again rubbed with quicksilver, without the aid of nitric acid, and is again ready for use. A copper plate, to be employed for this purpose, should not be too thin, since it would quickly become permeated by the quicksilver, and break almost as readily as glass; a plate of one-sixteenth of an inch in thickness will, however, with careful management, last several years. It is evident that the coarser gold will be caught near the head of the sluice, and the finer particles be arrested further down, in proportion to their state of division. When dirt contains a large proportion of coarse gold, the mercury is frequently introduced, at a distance of from forty to sixty yards from the head, so as to catch only the fine gold in the form of amalgam.

The collection of the dirt which accumulates in the bottom of the sluice, and the separation from it of the gold, amalgam, and quicksilver, is called the *cleaning up*, and the time between one cleaning up and another is called a *run*. In most cases, an ordinary sluice runs only during the day, but, in some instances, the work is continued throughout the whole twenty-four hours. A run commonly lasts about a week, and the cleaning up is not unfrequently reserved for the Sunday. This usually occupies about half a day, and consequently, on account of the loss of time entailed, must not be too frequently repeated. In some claims it is not done until the riffles are so worn and damaged, as to require to be overhauled and repaired.

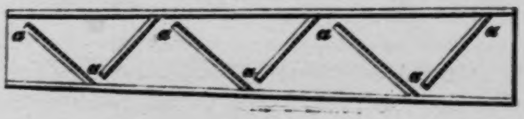
When a cleaning up has been decided on, no more dirt is thrown into the sluice, and the water is allowed to run through it until it passes off quite clear at the lower end. Some six or eight sets of bars are now taken up from the head of the sluice, and the dirt is washed down; whilst the gold and amalgam, which had been caught between them, is arrested by the first remaining set of bars, from whence it is taken out with a scoop into a pan or bucket. Another six or eight sets of riffle bars are now taken up, and the operation repeated, until the whole length of the sluice has been cleaned up.

The amalgam and mercury taken from the sluice are first separated from any admixture of sand by panning, and then strained, either through buckskin, or a piece of close canvas, which allows the liquid quicksilver to pass through, but retains the more solid amalgam. In order to obtain the gold in a metallic state, the amalgam must be subjected to a degree of heat capable of volatilising the quicksilver, which is thus expelled, leaving the gold in the form of a porous mass, of a light yellow colour. This operation is most economically performed in a cast iron retort, provided with a refrigerator, by which the mercury is condensed, and can be collected for subsequent use; but in many cases, and particularly in small claims, the miners drive off the quicksilver by heating the amalgam on a shovel, or on an iron plate, by which method of proceeding it is of course volatilised and lost, besides exposing those around to the danger of injury from the metallic vapours evolved. The amalgam obtained, after squeezing the superfluous mercury through cloth or buckskin, is generally calculated to afford from thirty-five to forty per cent. of retorted gold.

As before stated, the riffle bars are often between five and six feet in length, and sawn in the direction of the grain of the wood; but these, in claims where large quantities of pebbles and boulders are enclosed in the dirt, are rapidly worn away, and in some cases the bars are cut across the grain of the timber, and placed end upwards in the sluice. Riffles so constructed are found to last three times as long as those cut in the ordinary way, but it is generally difficult to get them more than about three feet in length, and consequently they require a little more time for fixing than the usual longitudinal bars. These *block riffles*, instead of being placed longitudinally in the boxes, are sometimes fixed transversely across them, and at distances of about two inches apart.

In small sluices, the bars are not always placed either longitudinally or transversely, but sometimes in a series of zigzags, as seen in Fig. 14.

FIG. 14.



ZIGZAG RIFFLES.

The first bar in the sluice is nailed at an angle of 45° with the course of the box itself, but does not touch the opposite side, between which and its extremity is left a space *a*, of about an inch in width. Immediately below this open space another bar is fixed, at right angles to the first, touching the side of the box beneath the opening, and stopping an inch short of the other side. This is continued until near the lower end of the sluice, where there are either longitudinal riffles, or transverse blocks, as before described. When this arrangement of the bars is employed, it is evident that, although a large proportion of the dirt and water will pass over them directly down the trough, the heavier particles will sink to the bottom of the boxes, and being directed by the oblique riffles, will assume a tortuous course through that portion of the sluice. At a short distance from the head of sluices of this description is placed a vessel containing mercury, in which is a small hole, from whence it is allowed, very slowly, to escape into the sluice. This quicksilver runs down the arrangement, following the course of the riffles; and overtaking, and coming in contact with, the particles of gold, unites with it, and forms an amalgam, which is retained in that portion of the sluice provided with ordinary longitudinal riffles. It is necessary that these, and all other descriptions of sluice, should be carefully watched, in order to prevent any jamming of the larger boulders, or any local accumulations of dirt, which would interfere with their efficient action, and result in loss of gold.

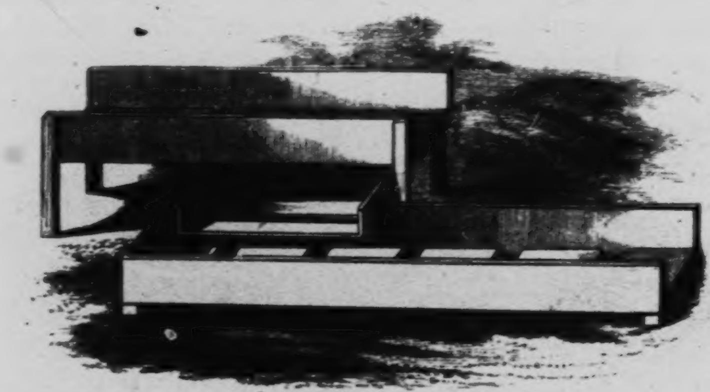
In small sluices, it is sometimes customary not to allow the large pebbles to run throughout their whole extent, and stones of more than a few pounds in weight are therefore thrown out by a man provided with a prong, having numerous blunt parallel teeth, which, without being liable to injure the wood of which the boxes are made, enables him to remove the stones. Another contrivance for collecting fine gold is to impregnate the wood of which the riffles are made, with mercury. This is done by means of an ordinary gas-pipe, which is first ground to a thin edge at one end, and then driven into the wood forming the block riffles. On subsequently filling this pipe with quicksilver, the pressure of the column forces it into the pores of the wood, which afterwards acts somewhat similarly to the amalgamated copper plates before described, except that the resulting amalgam is removed by simply scraping the surface of the blocks. This process is not, however, often adopted, and is not generally to be recommended, since it requires more time to prepare the blocks than to amalgamate an ordinary copper plate, besides which its action is, on the whole, not so satisfactory.

When two companies are working claims side by side, a double sluice is, for the sake of economy, sometimes employed. In this case the boxes are made of double the usual width, and are divided in the middle by a longitudinal partition, so as to form two distinct sluices. This arrangement is also employed in claims where, to prevent loss of time, one side of the trough is made use of, whilst the other is being cleaned up; or in localities in which the supply of water is, during a portion of the year, sufficient for working both sluices, whilst in the dry season the quantity at command is only enough for one.

The *under-current sluice*, another modification which is often found advantageous, is represented Fig. 15.

In this arrangement a grating is placed in the bottom of the lower extremity of the last box in the series, and beneath this is introduced another sluice with a lower grade and fresh supply of water. The impetus acquired by the large boulders, in the first sluice, causes them to roll off over the grating, and, together with a portion of the water, to escape at the lower end; whilst the introduction of clear water, the less inclination, and more moderate current, determine the arrest of many particles of gold that would, under ordinary circumstances, be carried off and lost.

FIG. 15.



UNDER-CURRENT SLUICE.

The rapid wearing of all kinds of wooden riffles, and the facilities they afford for stealing the auriferous amalgam by night, has led to the introduction of what are known as *rock sluices*, but these are attended with the disadvantage of being more difficult to clean up, and consequently to prepare for working after being cleaned up, than any of those before described. The sluice itself is formed of the ordinary rough wooden boxes, which are coarsely paved with lenticular rolled pebbles, laid on edge, and which, when the wash dirt has passed over them for a short time, become securely embedded by the sand which collects between them. In *tail sluices* the paving stones employed are generally larger than those used for the bottom of an ordinary sluice. These stone bottoms are only employed in wide boxes, and have generally an inclination of about an inch to the foot. When the sluice is cleaned up, the stones have necessarily to be removed, and the bottom of the boxes repaired before again commencing to wash; but this, with experienced hands, is performed with considerable rapidity, since a miner, accustomed to the business, will pave some 200 feet of an ordinary sluice in the course of a day.

Tail sluices are arrangements for collecting gold still retained by the dirt which has passed through the ordinary sluice, and escaped being thoroughly impoverished in its progress. They are usually placed in a ravine through which the tailings from one or more sluices of the ordinary construction flow, and are only cleaned up after the expiration of several weeks, receiving, in the meantime, no further attention than is necessary to prevent their choking. Tail sluices are generally large, long, and paved with blocks of stone, or wood placed on end, and often afford large profits to their proprietors. Both the dirt and water with which they are supplied is furnished by the sluices emptying into them, and they are in some instances made double, so that one side may be in working order whilst the other is being cleaned up. Many of the tail sluices on the Yuba are

Fig. 16.

Tail Sluices, Yuba River.
(*From a Photograph.*)

of extraordinary length, and some of them have been constructed as much as twenty feet in width. Fig. 16 is a view of two of the tail sluices on this river.

All the varieties of the sluice which have been hitherto described require wooden boxes or troughs, through which the water is conducted; but in localities where there is a large supply of water, plenty of pay dirt of low produce, and the necessary declivity, a sluice is sometimes improvised without the use of wood. Such arrangements are called *ground sluices*, and in order to prepare one, a small gutter is first made with a sufficient inclination through the dirt to be washed, and into this is directed a stream of water, by the action of which the channel rapidly becomes deepened and enlarged. As soon as the sides and bottom of this ditch have ceased to become rapidly eroded by the action of the current alone, the miners begin to assist the operation by pinching off scales of ground from the bank, which, falling into the stream, are acted on precisely as in the case of the ordinary board sluice. No mercury or riffles are employed in the ground sluice, but unless the bottom consists of a rough and irregular bed rock, a few large stones should be roughly thrown in for the purpose of arresting the gold, which, if the surface were not more or less uneven, would be liable to pass off and be lost. When a considerable amount of dirt has been passed through a sluice of this description, the water is diverted, and the auriferous matters, thus partially washed, collected, and cleaned up in a tom, cradle, or short box sluice. The ground sluice is seldom employed on claims where mining is continuously carried on, and is most advantageously introduced in localities in which water is abundant for a few days only, after heavy rains, and consequently where it would not be judicious to erect large and expensive sluices. When, as is sometimes the case, either in hydraulic or other mines, a sluice passes through an adit level, or day tunnel, it is called a *tunnel sluice*, but these differ in no respect from other sluices except in their situation.

River Mining.—River mining consists in turning the stream of a river, by means of a dam in connexion with a ditch or large wooden flume, and subsequently washing the dirt, found in its bed, for the gold which it may contain. The streams selected for this purpose are generally mere mountain torrents, of which the banks are steep and irregular; and consequently a ditch being impracticable, recourse is had to a rough wooden trough or flume. River mining can only be successfully carried on during the summer and early fall, when the water is not only low, but the miner will not be subject to have all his work destroyed by a sudden freshet, which might sweep before it his flume, dam, and all his tools together. This branch of mining is, however, subject to many disadvantages, from which the other descriptions are comparatively free. In the first place it cannot be carried on during more than half the year, and as the miner has no means of prospecting under the surface of the water, and thereby ascertaining the value of the dirt, it may, after he has expended a large amount of time and labour, prove almost worthless when the diversion of the stream has been effected. Secondly, he is constantly exposed to the danger of floods, which may, by carrying away his flume and destroying all his work, suddenly deprive him of the advantages to be derived from his outlay of capital and labour.

These disadvantages, coupled with the fact that the principal river washings have been already exhausted, have almost done away with this description of mining in California. In some few instances enterprises of this kind have been attended with highly remunerative results; but in the majority of cases, the gold extracted has not repaid the labour and capital expended to procure it. A long flume of a sufficient capacity to carry off the whole of the water of a considerable river, is in itself an expensive piece of work; but when it is taken into consideration that a greater or less amount of leakage from the dam has always to be contended with, and that water is constantly finding its way into the river bed from the various ravines trending in that direction, it becomes evident that the difficulties of river mining are of no ordinary character. It is therefore necessary to provide machinery for pumping out this influx of water; and as large rocks, often weighing several tons, are frequently met with, they have to be removed, by means of cranes, before the most valuable dirt can be reached. River mining is consequently never attempted by single individuals, but is always carried on by associated companies, either entirely composed of working miners, or by miners assisted by the storekeepers and other substantial inhabitants of the neighbourhood. In the latter case, those who are not directly connected with the business pay their proportion of the expenses in money, the merchants supply the necessary provisions, the saw-mill proprietors the wood, and the carpenters make the flumes, whilst the miners prepare the dam, and perform any other work that may be required.

Beach Mining.—Beach mining is the process of extracting gold from the sands on the sea-shore, and has been extensively carried on between Cape Mendocino in California, and the mouth of the river Umpqua in Oregon. The beach is here narrow, and lies at the foot of a bluff of auriferous sand, which, in stormy weather, is undermined by the waves, and the lighter constituents being washed away, leave sands which are often rich in gold. The gold is here found in a finely divided state, associated with the heavier and darker coloured sands, the position of which is frequently changed by the action of the tides and currents. In this way, a part of the beach which may, on one day, be deeply covered with sand, in which particles of fine gold can be readily seen, will, on the following, be left either bare, or covered with sand containing little or no gold, and it therefore requires constant attention on the part of those directing the operations, in order to select the most auriferous descriptions for treatment.

The Companies engaged in this kind of mining usually consist of about ten men, including the foreman, who every morning rides along the beach for a distance of about two miles, on either side of the camp, for the purpose of ascertaining where, on that day, the richest sands are to be met with. When this has been determined, each man of the company proceeds to the spot with two pack mules provided with raw hide sacks, *alforjas*, in which the sand is collected and carried to the washing place, which is sometimes several miles distant. Sea water is occasionally employed for this purpose, but fresh is generally preferred; and for this reason, the sand is in most cases transported to a running stream at no great distance from the shore. It sometimes happens that rich sand is not met with within twelve miles of the camp, but in such cases it requires to be highly auriferous in order to support the expenses of transport. The richest sands are found far down the beach at low tides, and consequently when such a tide occurs during still weather, every exertion is made to obtain as large a supply as possible; since with high tides and a rough sea, there is little to be done in this class of mining. The sands thus obtained, being entirely free from clay, are very readily treated, and a couple of days' working with a sluice will generally effect the washing of all the sands collected during a month.

Water Supply.—In proportion as the shallow placers in the ravines and river beds of California, from which the first supply of gold was obtained by simple means and at a small cost, became more or less completely exhausted, the necessity of devising some ready method by which the deep placers could be economically worked, naturally forced itself on the attention of the mining community. In order to accomplish this object, it was necessary to convey a copious supply of water to auriferous deposits far above the level of the rivers in the vicinity, and often situated at a great distance from the present streams of the country. This required the expenditure of large sums of money for the construction of the canals and aqueducts, by which it was brought from springs and reservoirs, at elevations such as to command the largest and most important diggings. This demand for the association of labour and capital soon called into existence numerous canal and ditch companies, the shareholders being for the most part miners, whose limited resources generally obliged them to borrow money from the local bankers, at rates of interest varying from three to five per cent. per month. In this way the Middle Yuba Canal Company was organised in 1853, the water being brought from a point on the Middle Yuba a little below Woolsey's Flat, where the river was dammed and the aqueduct commenced. The total expenditure involved in this enterprise, including the various branches, ditches, reservoirs, and extensions, has amounted to $600,000 or about 120,000*l.*; and the works are now capable of supplying thirty-eight cubic feet of water per second, which could at a comparatively small expense be increased to sixty cubic feet.

The first-named quantity is considered equivalent to nearly 1,500 inches, miner's measurement, and the second to 2,280 miner's inches.* The Eureka Canal was commenced in 1856, and has cost, including the various reservoirs and branches, $1,000,000, or 200,000*l.* The Eureka Lake is the largest reservoir now in connexion with this aqueduct, although it has been proposed to connect it with the Truckee Lake, which is a still more considerable body of water. At the outlet of the Eureka Lake a substantial granite dam raises its waters to the height of forty-two feet above their natural level. This is retained by a natural abutment of granite, capable of receiving, if necessary, an additional height of twenty feet. The width of this structure, at the base, is 120 feet, its height is seventy feet, and its length from bank to bank 250 feet. Its water face has a double lining of securely-fastened two and a half inch planking, and the flow of water is regulated by a sluice placed in a tunnel of strongly-arched masonry. The capacity of this reservoir is estimated at 933,000,000 cubic feet, or about five months' full supply of the canal. During from four to five months in each year, however, it obtains the necessary amount of water from other sources. The snow accumulates in the Sierras, in great quantities during the winter months, and the melting of this not only supplies the flow of the streams, but also fills, to overflowing, the various mountain lakes and artificial reservoirs, in which the waters are stored in reserve against the droughts of late summer and autumn. Besides this main reservoir, there are several others, of which Lake Faucherie is the most important.

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* The miner's inch of water, in California, is the quantity which will flow through an opening one inch square under a mean head of six inches, and the working day is generally calculated at ten hours.

The Eureka Canal is constructed partially of earth and partially of a wooden fluming: the dimensions of the main flume are five feet nine inches in width, and three feet in depth, with a fall of sixteen feet per mile* The discharge is ninety-six and a half cubic feet of water per second, or 3,667 miner's inches. We shall therefore be below the truth if we assume that this canal is capable of supplying, after allowing for leakage, 3,000 inches of water in a working day of ten hours, which is equal to 7,200 inches during the twenty-four hours, or, as the discharge per second is ninety-six and a half cubic feet, it will afford 8,337,600 cubic feet of water in twenty-four hours.

Among the most remarkable objects that strike the traveller on first visiting the mining regions of California, are the lofty aqueducts, constructed of trestle-work, for the purpose of conveying water across deep valleys and ravines. The Magenta and National aqueducts are the most considerable constructions of this description on the line of the Eureka Canal. The Magenta aqueduct is 1,400 feet in length, whilst the length of the National is 1,800 feet. The greatest height of the former is 126 feet, and that of the latter 65 feet. The dimensions of the flume are seven feet in width by fifteen inches in depth, and its inclination one foot in a hundred. In order, as much as possible, to avoid the action of the wind, which often blows strongly up the valley, the sides are made low, and considerable width given to the bottom. The legs of the trestles were all cut out of trees which grew in the vicinity, and are without a splice from foundation to top. The sides of the flume are made of whole scantlings, seven and a half inches in width, and thirty feet in length. The trestles are placed thirty feet from centre to centre, and are well and securely braced. This aqueduct was put together in sections of thirty feet, and each, when completed, raised into its place from the spot on which it was constructed, by which means the use of scaffolding was rendered unnecessary. The aggregate length of all the ditches belonging to the Eureka Company is about 200 miles.

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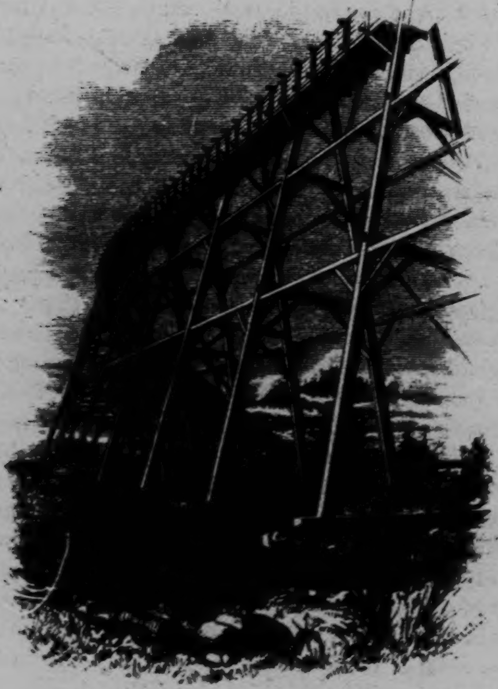
* When a less inclination is given, the water is liable to freeze in winter.

The foregoing description of the works of the Eureka Lake Water Company will afford some idea of the extensive scale on which such enterprises are conducted in California, as nearly all the more important mining districts now receive a plentiful supply of water by similar means. The following woodcut, Fig. 17, will serve to illustrate the method adopted for the construction of high aqueducts for mining purposes.

FIG. 17.



FLUME NEAR SMARTSVILLE, YUBA COUNTY.
(*From a Photograph.*)

Hydraulic Mining.—In order to treat most successfully the extensive beds of detritus forming the deep placers previously described, the following conditions are involved :—

1st. Whatever may be the depth of the auriferous gravel, the whole must be removed down to the bed rock.

2nd. This must, as far as possible, be effected by the force of a column of water, since manual labour becomes too expensive when from 1,500 to 4,500 cubic yards of dirt have to be disposed of during each working day of ten hours.

3rd. The mechanical disintegration of the more or less indurated gravel must go on contemporaneously with the washing of the resulting debris, and be effected by the same supply of water.

4th. Provision must be made for readily disposing of the large amounts of refuse resulting from the removal of such vast masses of auriferous gravel.

In practice these conditions are fulfilled in the following way :—

After having selected a sufficient extent of suitable ground, the water from a canal is brought, by side flumes or aqueducts, to the head of the mining ground, with an elevation of from 120 to 160 feet above the level of the bed rock, where it is conducted into a wooden tank, into which it constantly flows. This box is provided with a valve, and from it the water is conveyed to the bottom of the claim by means of a strong sheet iron rivetted pipe, from eight to fourteen inches in diameter, communicating at the bottom with a thick rectangular cast iron chamber, in the sides of which are apertures provided with slide valves and union joints, to which can be fitted strong flexible hose terminating in bronze nozzles from two and a half to three inches in diameter. The arrangement of the bulk head or pressure box, into which the water is conducted at the head of the column, will be understood by reference to Fig. 18.

The flexible hose are usually made of closely-sewn heavy duck, and will, without any external support, bear the pressure of a column of about fifty feet in perpendicular height; when however, as is most frequently the case, the pressure employed is greater than this, they require to be strengthened by iron rings. The bands employed are placed over the hose at intervals of three inches from each other, and connected by means of four longitudinal cords, dividing the circumference into equal divisions. These *crinoline hose* are very flexible, and will, if well made, support, without danger of bursting, the pressure of a column of water a hundred and eighty feet in height. Instead of increasing the strength of the canvas hose by the use of iron rings, they are sometimes tightly covered with a netting of cord, half an inch in diameter, forming meshes two inches square. In some claims also the cast-iron chamber at the bottom of the pressure pipe is dispensed with, and a separate iron pipe connected with each nozzle, or two or more hose joined to each by means of T pieces; but the use of one large pressure pipe with the close chamber at the bottom is, in most cases, to be preferred. From each of these nozzles streams of water are directed against the face of the gravel to be washed, with a force which can only be compared with that of ordnance, and the power developed by this means, where the supply of water is large and the height of the column considerable, is perfectly astonishing. The volume of water employed varies in accordance with various local circumstances; but it is not uncommon to see four or five such streams acting simultaneously against the face of the same bank—five hundred miner's inches of water constantly discharged against the face of a bank, under a pressure of from sixty to a hundred pounds to the square inch, aided by its disintegrating and softening action, bring down large sections of the gravelly mass,

FIG. 18.

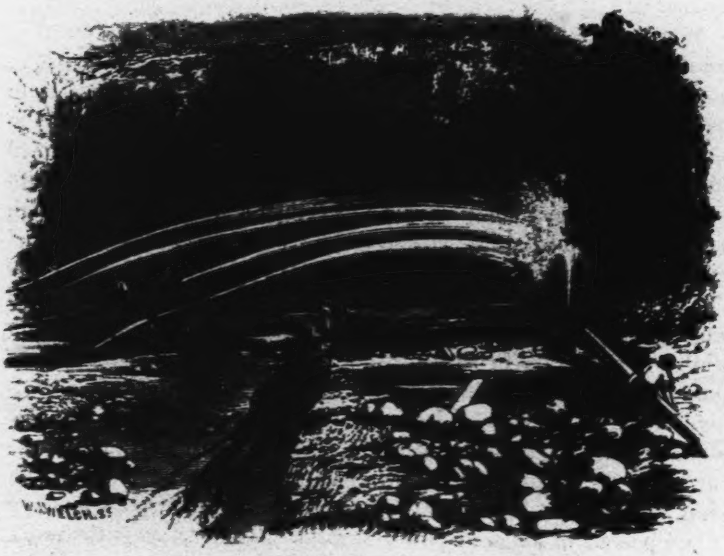
PRESSURE BOX, YUBA RIVER.
(*From a Photograph.*)

which fall with violence, and render it necessary for the workmen directing the operation to exercise great caution in order to avoid accidents.

The number of men necessary to carry on the operations of a hydraulic claim is very limited, since, in addition to those engaged in directing the nozzles, only one person is generally employed in attending to the sluice, so as to prevent its being choked by the dirt washed down from above.

The following woodcut, Fig. 19, shows the method of applying the jets of water against the side of a bank of dirt.

Fig. 19.



Hydraulic Mining, Washing down Bank.

(*From a Photograph.*)

The debris thus produced becomes rapidly disintegrated, and, borne along by the resistless force of the water, is carried forward to the sluice, through which it passes with the whole volume of the turbid stream.

Banks of more than eighty feet in height are generally worked in two benches. The upper half is never so rich as the lower, but is usually less compact and more easily removed by the action of water. The lower section, on the contrary, is often very closely cemented together, and most frequently requires the aid of gunpowder in order to loosen it, so as to enable the water to remove it with sufficient rapidity. For this purpose a tunnel is driven into the bank, at the level of the bed rock, for a distance of some fifty or sixty feet, and from its extremity another drift is extended on either side, at right angles, in which a large quantity of gunpowder is placed. The charge usually varies from fifty to two hundred barrels, which after being securely built in, is fired by a slow match, and by its explosion loosens a large mass of the compact conglomerate, which is afterwards readily acted on by the force of the currents of water brought to bear against it.

When the conformation of the country admits of it, a tunnel is sometimes brought in from the nearest and most convenient ravine, at a considerable depth in the bed rock, at a gradient of from one in twelve to one in twenty, and varying in length from a few hundred to several thousand feet. Such tunnels have not unfrequently occupied from three to four years in driving, and have cost very large sums of money. On the extremity of this long tunnel a shaft is sunk, through which the pay dirt is washed, and the level itself becomes a channel in which the sluice boxes are fixed, for the double purpose of directing the stream and collecting the gold. In such cases it is usual to employ a double sluice, in order that one side may be cleaned up whilst the other is in active operation, by which means all loss of time in working the claim is avoided. When this method of proceeding is adopted, the debris produced by the united action of the jets of water directed against the bank is rapidly carried forward by the current to the mouth of the shaft, down which it is precipitated with great violence, and being often accompanied by boulders of a hundred or two pounds in weight, a powerful disintegrating action is the result of the fall.

In some claims a system of tunnels is extended in the bed rock, very much as in a coal mine; and after dividing the ground into separate blocks by proper levels, they are washed away by the hydraulic process, after which the whole mass settles down and is easily disintegrated by the action of water.

The sluices employed in connexion with hydraulic mining are made wider than those used for other purposes, and are sometimes provided with wooden riffles, kept apart by slips of wood; in others their bottoms are paved with stone, as previously described. When wooden riffles are employed, they are composed of blocks of wood cut transversely across the tree, and placed end upwards in the sluice box. In this case the cavities which occur between the different blocks form the spaces in which the gold is collected, and are found very efficient for this purpose. A good general idea of the appearance of the sluice belonging to an ordinary hydraulic claim is afforded by Fig. 20.

FIG. 20.

SLUICE AND TUNNEL, TIMBUCTOO.

(*From a Photograph.*)

Rude as this method of saving gold appears, experience has shown that a larger proportion of the metal is collected by it than by any other process, and that at the same time the cost of handling a cubic yard of dirt is infinitely less by the hydraulic, than by any other system of mining. In fact, it would be utterly impossible to treat such enormous masses of dirt, as are now daily operated on, by any other known means.

As an illustration of the amount of work which can be performed in a given time by hydraulic mining, we give the results obtained at the Eureka Claim, near San Juan, California, where the bed of pay dirt is about 135 feet in depth.*

The upper portion of this deposit to the depth of seventy feet does not contain a large amount of gold, but is easily washed, whilst the lower portion, having a thickness of sixty-five feet, is much richer, but cemented together, and the work is therefore carried on under conditions of considerable difficulty. The pay dirt is reached by a bed-rock tunnel of a great length, which cost on an average $8 per foot, and of which the total expense was $28,000. The work is carried on by means of four *jets d'eau*, discharging together about 208 gallons per second, or 12,500 gallons per minute, under a pressure of 140 feet. The whole of the operations are conducted by four men, and after the expiration of ten working days, the washing down of fresh earth is suspended and the sluices cleaned up.

During this period of ten days about 36,500 cubic yards of gravel are worked over, the cost of working being nearly as follows:

Cost of Water	$1,000
Labour	173
Sundries	100
Total	$1,273

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* The following statistics relate to another hydraulic claim near the same locality:—Capital invested, $7,000. Length of time during which the claim has been worked, 16 months, in two seasons of 8 months each. Water used, 350 miner's inches; pressure, 160 feet: water brought 2,800 feet, through 13-inch pipe; used through two nozzles of 2½ inches, and one nozzle of 2 inches diameter; water all used through the pipes. Running night and day; twenty-four hours estimated as a day's work; water costs 20c. per inch, or $70 per day. Total expenses per day, about $115. During the two seasons this company has paid all its expenses, and repaid $4,000 of its capital. The gravel has averaged, gross returns, about 3c. per ton of 15 cubic feet; the quantity removed daily has been about 4,250 tons. If this gravel had been even slightly cemented, the cost of working would have been materially increased. The above are supposed to be about the poorest gravels worked in California.

The gold taken from the sluice at the end of this period fetches, on an average, $6,000; and when the operations are confined to the lower portions only of the gravel, the value of the gold obtained increases from $16,000 to $20,000.

As an evidence of the enormous advantages possessed by the hydraulic process over every other system of placer mining, it may be stated, that taking a miner's wages at $4 per day, the cost of handling a cubic yard of gravel will be nearly as follows:—

With the pan	$20·00
" rocker	5·00
" long tom	1·00
By hydraulic process	·05

With reference to the enormous power of this system, Professor Silliman observes, "Man has in the hydraulic process taken command of Nature's agencies, employing them for his own benefit, and compelling her to surrender the treasure locked up in the auriferous gravel by the use of the same forces which she employed in distributing it." *

The water is usually supplied to the various mining claims by independent associations, who charge for it at rates varying from 10c. to 40c. per miner's inch per working day of ten hours, but the average charge may be estimated at 16c. The consumption of each mining claim in active work may be taken at about three hundred miner's inches; and this quantity flowing for ten hours will be equal to 284,210 cubic feet, or 1,771,219 imperial gallons. †

The amount of water distributed to the various claims is ascertained by conducting a stream from the canal into a measuring box, from twelve to fourteen feet square, in the sides of which openings are made two inches in height, and extending nearly across three of its sides. These openings are closed by slide valves when not required,

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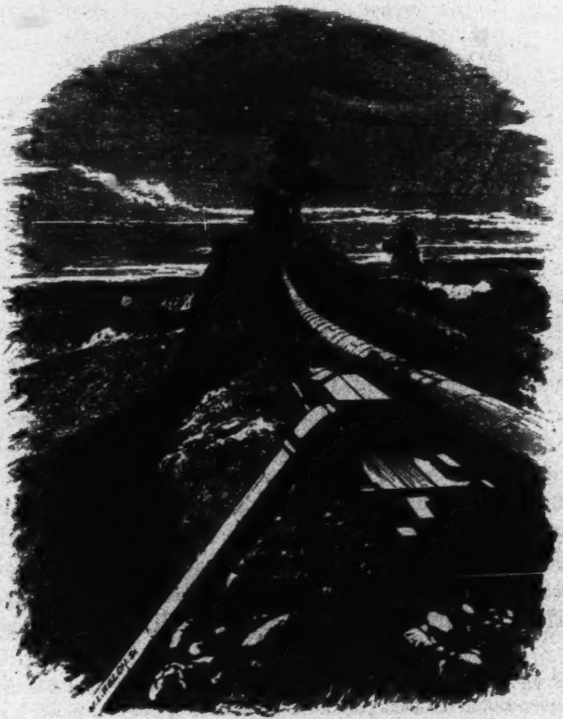
* Report on the Deep Placers of the North and Middle Yuba, p. 22. Silliman.

† In order to soften the bank, when the gravels are slightly cemented, it is usual to cut a trench across it, a short distance in advance of the breast. Water is allowed to flow into this, and, by percolating through the gravels, it has a tendency to soften the cementing ingredients of the deposit. When the bank is naturally soft, and consequently removed with great facility, it is customary to admit a certain amount of water into the sluice in addition to that passing through the nozzles. This is done for the purpose of so diluting the clay and fine sand as to allow the gold an opportunity of coming in contact with the mercury between the riffle blocks.

and the sectional area through which the water flows determines the number of inches used. The amount of water discharged by one miner's inch, in ten hours, is equivalent to 947 cubic feet.

In some cases, instead of bringing the water directly into a claim by means of an open aqueduct, it is found more convenient to place the pressure box on some neighbouring elevation, from which the supply is conducted, by an inclined iron pipe, to the hose box at the face of the diggings. Fig. 21 represents this method of arrangement.

Fig. 21.



Iron Pipes.
(From a Photograph.)

The iron pipe, although, in California, generally more expensive than a wooden flume, has the advantage of preventing loss by evaporation and leakage, whilst the full pressure can be obtained at any point along its course by the introduction of a union joint, to which a valve may be attached, for the purpose, when necessary, of cutting off the supply; such a branch and valve are seen in the woodcut to the left of the main pipe. It sometimes happens that the pay dirt in a hydraulic claim contains a thick band of non-auriferous clay, which is not readily taken up by water, and is liable to pick up gold if allowed to pass through the sluices in the form of rolled pellets. In such cases the clay band is undermined by the judicious use of the nozzles, and, after dividing the detached portions into fragments of a manageable size, they are taken up by a crane provided with a long jib. This, by making half a revolution, deposits the mass behind the workings, and at a distance of many yards from the face of the pay dirt; after the removal of the clay, the washing of the dirt proceeds in the usual way. This arrangement is also used for the removal of any very large boulders that may be met with in the dirt.

Cranes used for this purpose are frequently worked by hydraulic pressure supplied by a branch pipe from the main, and are very simply constructed. To the pinion shaft of the winch is attached a S-shaped flyer, through which the water rushes, as in Barker's mill, and a rotary motion is the result. The orifices of the revolving fly are provided with valves so constructed, that, in case of the speed becoming too great, they become partially closed by the centrifugal force acquired, and the action of the machine is consequently as easily controlled as that of the steam-engine by its governor. Companies formed in California for the purpose of supplying the various diggings with water have, as a general rule, been highly remunerative to the shareholders, and the large annual revenue derived from some of them will be understood when it is stated that the receipts of the Eureka Company alone, during three years, amounted to $626,560 (= 125,312*l.*). The expenses during the same period were about $120,000, or 24,000*l.*, leaving a clear profit of $506,560, or 101,312*l.* on the three years' operations.

Dry Washing.—Dry Washing is a process for winnowing gold from dirt, which in the early days of Californian mining was much employed, by Mexicans, in localities where water could not be procured for the purpose of washing. In diggings of this description the richest dirt only is selected and placed on a raw hide, on which, after it has become thoroughly dry, it is pulverised by rubbing between the hands, and the coarser particles of stone and pebbles picked out. The miner now takes a large shallow basin, called a *batea*, and with a circular motion throws the dirt into the air, allowing the wind to carry off the lighter portions, and catching the remainder as it falls in the batea. This operation is repeated until what remains in the basin contains a large proportion of gold, when its further purification is effected by blowing with the mouth. This operation is similar to the old method of separating corn from chaff, and is conducted in very much the same way. Instead of using the wooden dish, two men sometimes take a hide, or blanket, in which they throw up the dirt for the purpose of exposing it to the action of the wind. The dry digger never goes very deep for his dirt, but unless it be rich in gold this method of operating cannot yield him remunerative returns. Instead of removing the surface earth, he frequently digs a pit six or eight feet deep, and then burrows after the pay dirt. This method of mining, which is not entirely confined to dry washings, is called *coyoting*, from the supposed resemblance of openings so made to the burrows of the coyote, or Californian wolf.

CHAPTER IX.

VEIN MINING IN CALIFORNIA AND AUSTRALIA.

QUARTZ MINING—TESTING GOLD QUARTZ—BATEA—HORN SPOON—ARRASTRE—CHILIAN MILL—STAMPING MILL—AMALGAMATING IN BATTERY—IRON BATTERY BOX—SCREENS—SINGLE CAMS—SEPARATION OF GOLD—BLANKETS—AMALGAMATED COPPER PLATES—CLEANING UP—SETTLING PITS—AMALGAMATOR—ATTWOOD'S SYSTEM OF AMALGAMATION—LOSS OF GOLD—CONCENTRATION OF TAILINGS—ROCKER—CONCAVE BUDDLE—BRADFORD'S SEPARATOR—EXTRACTION OF GOLD FROM SULPHIDES—BAUX AND GUIOD'S AMALGAMATOR—CHLORINATION PROCESS—ANALYSES OF CALIFORNIAN PYRITES—RETORTING—MELTING—TABULAR STATEMENT OF THE OPERATIONS OF THE PRINCIPAL CALIFORNIAN QUARTZ MILLS.

THE processes employed for the extraction of gold quartz differ in no respect from ordinary mining operations applied to the working of regular mineral veins, and will consequently require no special description. When the conformation of the country admits of working by means of a day level, it is driven in from some convenient valley in the neighbourhood of the vein, and the rock is obtained by stoping in the usual way. If, on the contrary, there be no facilities for this method of commencing the work, shafts are at once begun from the surface, either perpendicularly, so as to intersect the lode at some convenient depth, or, still more frequently, an inclined shaft is put down on the course of the vein itself. When the mine is worked by means of an adit level, the rock broken in the interior is trammed through it into the open air for subsequent treatment; but if the extraction be conducted by the aid of shafts, the auriferous quartz is drawn to the surface by skips or waggons running on inclined tramways in connexion with a horse whim or steam-engine. In the case of very narrow veins some of the enclosing rock has frequently to be broken with the lode in order to afford room for the miner, whilst, in wide ones, it often happens that a portion only is sufficiently rich to pay the expenses of extraction and treatment, and the remainder is consequently allowed to remain in the mine. Sometimes also the walls of a vein, and particularly the foot wall, contain a sufficient amount of gold to make its extraction a matter of importance, and when this occurs a portion of the enclosing rock is necessarily excavated. In some veins the rock is, generally speaking, barren, containing traces merely of the precious metal, the gold only occurring in paying quantities in pockets, at considerable distances from each other. In others the pay rock forms bands or streaks, running more or less parallel with the walls of the lode, and frequently separated from the non-productive portion by a distinct heading or band of country rock. Other veins are productive throughout their entire width, but seldom contain visible gold. These veins are not unfrequently lamellar in their structure, and contain thin interfoliations of slate parallel with their walls, as in the case of the Norambagua lode in Grass Valley. The most profitable leads are usually those which afford a large supply of rock obtainable at a cheap rate, and uniformly yielding an amount of gold in excess of the cost of extraction and treatment.

In all mines, however, the produce is, to a certain extent, variable, and in the majority of cases the best rock occurs in shoots having a known inclination in the direction of the extension of the vein in length.*

From the general irregularity of the produce, it is impossible to ascertain the average yield of vein-stuff without crushing and experimenting on large quantities; but the most usual method of judging, approximately, of the value of rock, is to pulverise a small quantity and wash the resulting powder in a batea or horn spoon. In selecting the rock for this purpose, it is evidently of the greatest importance that it should represent a fair average of the vein or streak from which it is taken, and consequently several hundred-weights should be broken from the whole area of the exposed surface, taking care that every part be represented by samples of nearly equal weights. The whole must now be broken by a hammer on an iron plate, into pieces of about the size of walnuts. The resulting heap is then carefully mixed, by turning over with a shovel, and subsequently cut through the middle, so as to leave a trench through its centre, extending to the floor on which it has been placed. The two sides are afterwards carefully scraped down, and removed as a representative sample on which the yield of the vein is to be estimated. For the purpose of a rough approximation this may be at once pulverised in a mortar or otherwise, and its contents judged of in accordance with the results obtained by washing. Where, however, greater accuracy is aimed at, and the original heap contained a large quantity of broken rock, at least a hundred-weight should be scraped down from the sides of the cutting, and this, after being further reduced to the size of peas, must be again cut through, and a sample of about four pounds obtained, by the means employed in the first instance, as the final sample. This is pulverised in a mortar, and the whole of it passed through a sieve of wire gauze, of forty holes to the lineal inch, after which it is ready for treatment, either by washing or assay.

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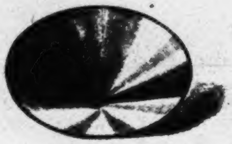
* These pay shoots have generally the direction of the striæ formed by dynamic action on the walls of the vein.

The most accurate results are obtained by carefully washing a four-pound sample in the batea (Fig. 22), which is about twenty inches in diameter, and two and a half in depth.

Fig. 22.



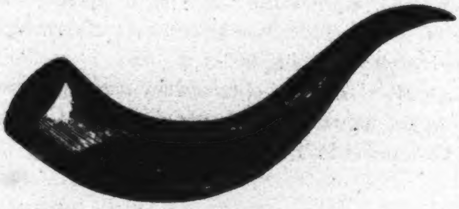
Batea.

After having in this way concentrated the gold in about an ounce of sand and pyrites, this residue may be either subjected to assay, or the sulphides dissolved by nitric acid, and the gold extracted by amalgamation with a little mercury, which is subsequently volatilised, and the gold weighed. In either case, calculations are made on the four-pound sample, and when the residue has been subjected to fusion very accurate results are obtained. When amalgamation of the residue is resorted to, allowance must be made for the increase of weight arising from the impurities usually contained in the gold of the district.

The spoon employed for washing small samples of pulverised quartz is made by cutting a large ox horn as shown at Fig. 23, and is generally about three inches in width and eight or ten in length.

In this spoon about a pound of the powdered quartz is carefully washed, and from the results obtained the miner is enabled, by practice, to judge, with a certain degree of accuracy, as to the probable yield of the rock. The quartz, after being broken from the vein, must be finely pulverised before the extraction of the gold it contains can be effected. Various contrivances are used for this purpose, but one of the simplest forms of apparatus employed is the arrastre.

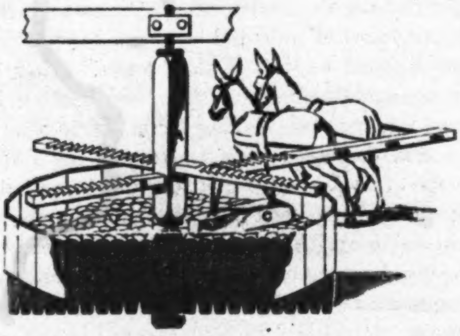
FIG. 23.



HORN, SHOWING METHOD OF CUTTING SPOON.

Arrastre.—The arrastre consists of a circular pavement of stone, about twelve feet in diameter, on which the quartz is ground by means of two or more large stones, or mullers, dragged continually over its surface, either by horses or mules, but more frequently by the latter. The periphery of the circular pavement is surrounded by a rough kerbing of wood or flat stones, forming a kind of tub about two feet in depth, and in its centre is a stout wooden post, firmly bedded in the ground, and standing nearly level with the exterior kerbing. Working on an iron pivot in this central post is a strong upright wooden shaft, secured at its upper extremity to a horizontal beam by another journal, which is often merely a prolongation of the shaft itself. This upright shaft is crossed at right angles by two strong pieces of wood, forming four arms, of which one is made sufficiently long to admit of attaching two mules for working the machine. The grinding is performed by four large blocks of hard stone, usually porphyry or granite, attached to the arms, either by chains or thongs of raw hide, in such a way that their edges, in the direction of their motion, are raised about an inch from the stone pavement, whilst the other side trails upon it. These stones each weigh from three to four hundred pounds, and in some arrastres two only are employed; in which case, a single mule is sufficient to work the machine. The following woodcut, Fig. 24, is a sectional view of a Mexican arrastre, as usually constructed; in which A is the upright shaft; B, arms to which mullers C are attached; and D, the central block of wood in which the lower bearing works.

FIG. 24.



ARRASTRE.

Some of the arrastres used by Mexican gold miners, and for the purpose of testing the value of quartz veins, are very rudely put together, the bottom being made of unhewn flat stones laid down in clay; but in a well-constructed arrastre, intended to be permanently employed, the stones are carefully dressed and closely jointed, and, after being placed in their respective positions, are grouted in with hydraulic cement.

The charge for an ordinary arrastre is four hundred and fifty pounds of quartz, previously broken into pieces of about the size of pigeons' eggs. The machine is now set in motion, a little water being from time to time added, and at the expiration of from four to five hours the quartz has become reduced to a finely-divided state, and more water is added, until the contents of the arrastre assume the consistency of tolerably thick cream. Quicksilver is then sprinkled over its surface to the amount of one ounce and a quarter for every ounce of gold supposed to be contained in the finely-divided rock, which is generally known, with a considerable degree of accuracy, from the results obtained from previous charges. The grinding is after this continued for another two hours, during which time the mercury is divided into minute globules, and becomes disseminated throughout the mass, which should be of such a consistency as not to allow it to sink to the bottom, but be so held in suspension as to meet, and amalgamate with, all the particles of gold. At the expiration of this time the amalgamation is considered complete, and the process of settling the amalgam from the ground silicious matter is commenced. Water is now let into the paste so as to render it very thin, and perfectly mobile, the mules being driven very slowly, in order to allow the particles of gold and amalgam to yield to the influence of their densities, and to sink to the bottom. After having in this way slowly agitated the mixture for about half an hour, the thin mud is allowed to run off, leaving behind it, in the bottom of the arrastre, the gold combined with mercury in the form of amalgam. Another charge of broken quartz is now put in, and the operation is repeated, time after time, until it is thought desirable to stop for the purpose of cleaning up. The length of a run, or the period which is allowed to elapse between one cleaning up and another, varies according to circumstances. In the roughly constructed arrastre, having a bottom of uncut stones laid in clay, the run is seldom less than ten days, and is sometimes extended to three weeks or a month. In this case the amalgam settles in the crevices between the paving stones, which have to be dug up, and all the sand and mud between them carefully washed. If, however, the machine be well constructed, and provided with a closely-paved bottom, the cleaning up is more frequently repeated, since the quicksilver and amalgam do not find their way so readily between the stones, but remain on the surface, from which they are easily collected in an iron vessel, for subsequent treatment by straining and retorting.

The arrastre does its work slowly, and consumes a large amount of power in proportion to the quantity of rock crushed, but is an excellent amalgamator, and is often valuable for the purpose of testing newly-discovered veins, and ascertaining their approximate yield. It is also the arrangement most commonly adopted by a miner, who, having found a rich pocket in his vein, is desirous of converting a portion of it into money, and of ascertaining whether it be likely to continue productive, before incurring the expense of erecting more costly and complicated apparatus. A modification of the arrastre is not unfrequently employed for the treatment of pyrites separated from tailings by washing, and is generally considered to be well adapted for that purpose.

The Chilian Mill.—The Chilian Mill consists of a vertical runner, frequently of granite, revolving on a horizontal arm projecting from a perpendicular shaft, to which motion is given either by water or steam power, or by being attached to horse gearing. The basin in which the runner revolves is usually slightly conical, and may be made either of stone or cast iron. In this arrangement the grinding area is regulated by the difference of the circumference of the circles described on the bed stone, by the inner and outer edges of the runner. In the majority of cases the Chilian mill, instead of having only one runner, has two, one on each side of the vertical shaft; and in such mills they are fastened at different radial distances from it. The method of working rock by the Chilian mill is very much like that employed with the arrastre, but the former is now seldom used, as the latter is generally considered to be a more efficient machine.

In order to show how expensive and inefficient this method of crushing is, we quote the following particulars relative to a Chilian mill, employed at the Silver Works of Constante, in Spain, given by Mr. Darlington, in an article on crushing and grinding machinery, contributed by him to "Ure's Dictionary:"—

Diameter of edge runner	6 feet.
Width „ „	centre, 20 in.; edge, 16 in.
Weight „ „	3 tons, 15 cwt.
Speed „ „	200 feet per minute.
Diameter of interior circle	4 feet.
Size of stuff previous to its being ground .	10 holes to the lineal inch.
„ „ after its leaving the mill . .	60 „ „ „
Quantity of stuff reduced per 10 hours . . .	350 lbs.
Horse-power employed	7

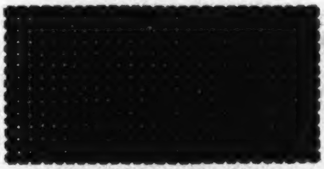
Stamping Mill.—In addition to the arrastre and Chilian mill, there are various other contrivances for the reduction of ores, and the extraction of the gold they contain; but although a vast amount of ingenuity has of late years been expended on this subject, none of the numerous modern machines can in any way compare in efficiency with the ordinary stamping mill. This may be said to be the only apparatus extensively employed for the reduction of auriferous rock in all parts of the world, and, in spite of local peculiarities, differs but little in its arrangements from the same machine, as applied to the crushing of tin and lead ores in various parts of Europe. The stamping mill essentially consists of a series of heavy pestles, working in a rectangular mortar, each of which is alternately lifted by means of a cam, and subsequently let fall with its full weight upon the ore to be operated on, and of which, after being previously reduced to fragments of proper dimensions, a constant supply is introduced into the mortar, or battery box.

When quartz mining was first practised in California, the lifters or stems of the pestles employed were made of wood, furnished with cast iron heads, attached by means of a wrought iron shank driven into the lifter, and secured by two strong rectangular bands of flat iron. In most mining districts in which these wooden stems are used, the lifting of the pestle is effected by a large wooden or cast iron drum, around the periphery of which cams are arranged in a spiral form, which, coming in contact with tongues, or tappets, fixed in the lifters, they are raised to a certain height, and, being suddenly released by the continuous motion of the axle, fall with their whole weight on whatever may happen to be beneath them. In California, however, another arrangement is employed for imparting motion to the pestles or stampers of a battery with wooden stems. Instead of a large cylindrical axle, a wrought iron shaft is made use of, and on this are keyed a series of long curved cams, which enter mortise or slot holes, in the several stems, and cause them to be alternately lifted and released, precisely as in the case of the ordinary stamping mill, provided with tappets and a drum axle. When wooden stems are made use of, they are usually about six inches square, and cut out of ash, or some other hard wood, having a straight grain. These wooden stems with square heads have, however, been almost universally superseded by the rotary stamp, with a round stem of iron, to which a circular motion is given by the friction of the cam in lifting, and which, being continued up to the moment of its release, is prolonged during its descent, thus imparting a grinding action to the cylindrical head at the moment of its coming in contact with the rock to be broken.

The rotary stamp is said to be more efficient than the rectangular one, and to grind a larger quantity of rock in a given time ; but however this may be, it is certain that the faces of the heads wear more evenly, and that a rotating battery requires less frequent repairs, than one made on the old principle. The battery box is generally composed of one solid casting, and usually receives either four or five stampers; when additional reducing power is required, other similar boxes are placed on the same line. In most instances, such batteries are arranged in sets of five stampers in each mortar, two of which are placed side by side in the same framing, ten stampers being thus set in motion by one shaft. Two five-stamp batteries, of a construction frequently employed, are represented Plate II., fig. 1 being a back elevation, and Fig. 2 a transverse section; the iron rods A are the stamp stems, B the shaft, and C the cams. This shaft is provided at one end with a large pulley D, which is generally constructed of either kiln-dried wood on arms, inserted in a cast iron boss, and then turned off in place, or is built solid of well-seasoned planks on a bored boss, and, as in the other case, turned, after being keyed to the shaft. When several of these batteries are arranged in one house, the motive power is communicated, by means of a broad belt, to the intermediate shaft B′, which is fitted with pulleys corresponding to those on the shafts B, with which they are severally connected by belts. These belts, which are manufactured out of a combination of canvas and india-rubber, are, from the first motion to the intermediate shaft, sometimes as much as two feet in width. The belts from the second motion to the shaft on which the cams are keyed, are made of a thinner material, and are from a foot to fourteen inches wide. The lift of the stampers varies from nine to twelve inches, but ten inches may be considered as about the average, and their weight, including the iron stem, varies from 550 to 900 lbs. The order in which the several stampers, included in one box, strike their blows, in a five-stamp battery, is not always the same in all establishments, but in most instances the first blow is struck by the central stamp. This is followed by the outside one to the right, then by the second to the left, afterwards by the second to the right, and finally by the stamper on the extreme left of the series. The number of blows struck by each stamper is from sixty to eighty per minute. The first portion of the stamper *a* is sometimes cast on to the stem, but more frequently it is fastened by wedges, and has a round aperture, in which is inserted the spill of the shoe *a′* firmly driven in or fastened by dry wooden wedges, which expand on coming in contact with water, and hold it securely in its place. The battery box is either of iron, cast in one piece, or its bottom alone may be of cast iron, and its sides of wood; in which case the lower portion of it, together with the inside of the feed hopper, must be lined with sheet iron, half an inch in thickness, fastened by five-eighth bolts. Immediately under each of the stampers is placed a short cylinder of cast iron *a″*, which is retained in its position, either by fitting into a circular bedding, in which it may be keyed by wooden wedges, or it is provided with a square flange, which, coming in contact with those of the other dies and the sides of the box, act as distance-pieces, by which it is kept in its proper position. These, and the shoes of the stampers, are, when worn out, readily replaced by new, a considerable economy of time and money being the result; the parts worn out are merely coarse castings of chilled iron, without any kind of fitting. The hole *x* is for the purpose of forcing in a drift above the spill of the shoe, and thus removing it when a new one is required, whilst the hole *x'* is employed in the same way for getting off the boss from the stem. In Grass Valley, and some of the other more important mining districts, the boxes E are, almost without exception, composed of single iron castings; but in localities where amalgamation is conducted in the battery itself, the sides and ends are sometimes of wood, the bed alone being made of iron; and when this method of construction is adopted, two plates of amalgamated copper, one-eighth of an inch in thickness, are often bolted at *b* on either side of the row of stampers. The rock to be crushed is introduced by a shovel at *c*, and a plate of perforated sheet iron, fastened either in a wooden frame, or retained in its place between the two rectangular iron bands, tightened by cotters, is introduced before the opening *d*. The battery bed, whether entirely of iron, or consisting only of an iron bottom, with lined wooden sides, is firmly bolted to a block of wood, at least two feet square, and of which the dimensions, when very heavy stampers are employed, are even much greater. This either forms, as in the drawing, a portion of the general framing of the arrangement, or is now more frequently, to prevent jarring, made quite independent of it. It is, however, essential that this portion of the structure should be well bedded on a solid foundation, and, if possible, rest directly on the bed rock. Occasionally quartz is crushed dry, but much more frequently water is admitted, and for this purpose a gas-pipe *e* affords the necessary supply, which enters the boxes through the branch pipes *f*, fitted with cocks for regulating the quantity introduced. The studs *g*, against which the cams come in contact, and by which the stampers are raised, are fitted on the iron stems by means of keys, which admit of their positions being readily shifted, when rendered necessary by the wearing of the shoes.

The props *h* shown in one of the batteries, but omitted in the other to avoid complication, are used for keeping up the stampers, either when the battery box is being cleaned, or when a portion of the machinery is thrown out of action for the purpose of repair; to do this they are successively pushed forward, so as to catch beneath the several bosses *g*, when lifted by the cams to their full height. When not so employed the props are allowed to fall back out of the way of the stems, as shown in the sectional drawing. The size of the apertures in the gratings or sieves at *d*, differ in accordance with the fineness of the gold contained in the rock treated, and is also, to a certain extent, varied in conformity with the particular views of the superintendent of the mill on that subject; but it is evident that with very small apertures the amount of quartz crushed, all other conditions being equal, will be less considerable than when a coarse screen is employed. The size of grating commonly made use of in some of the best mills in the Grass Valley district, is shown Fig. 25, which is known as "No 8."

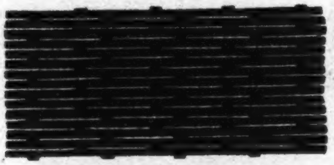
FIG. 25.



STAMP GRATE, WITH ROUND HOLES.
(*Full Size of Apertures.*)

In order to combine strength with the largest possible open surface, the apertures are sometimes made of an oblong form, and arranged as in Fig. 26.

FIG. 26.



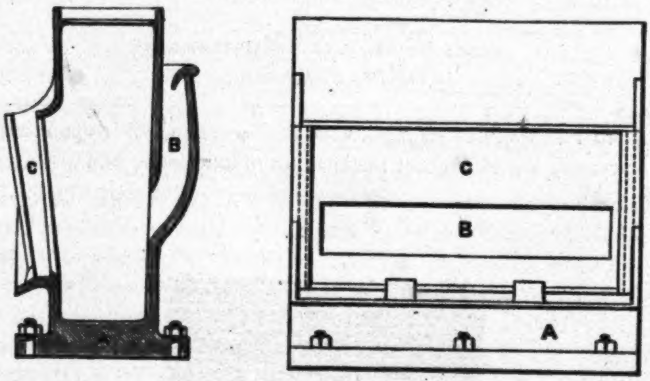
STAMP GRATE, WITH OBLONG HOLES.
(*Full Size of Apertures.*)

In some establishments these gratings are fixed perpendicularly, as seen in Plate II.; but more generally they are slightly inclined outward, and this arrangement is evidently attended with certain advantages.

When the grating is placed perpendicularly, a particle of quartz or other pulverised matter, splashed against the screen by the fall of the stampers, can only pass through it in case of being projected directly through one of the openings, and should it strike against a portion of the solid plate between the holes, will run down with the water on the inside, and again settle in the battery box. If, on the contrary, the grating be placed at a considerable inclination outward, as shown Plates III. IV. and VIII., a particle of pulverised rock, which has not been projected immediately through the grate, may, on running back with the water over its inner surface, pass through one of the apertures and escape into the trough on the outside of the battery.

In all machines of this description, it is of importance that each particle of the rock operated upon, should escape from the action of the stampers, as soon as it has become sufficiently reduced in size, and with this view the grate surface is in the Californian mills extended as much as possible, being generally made of nearly the full length of the battery box. With a view of supporting the grating, and protecting it against injury from the impact of the water dashed against it by the falling stampers, the sheet iron plate is externally strengthened by the application against it of some thin iron bars.

FIG. 27. FIG. 28.

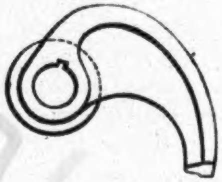


IRON BATTERY BOX.

When the high cast iron mortar is made use of, which is now generally the case, it has the form represented Figs. 27 and 28, which the first is a transverse section, and the second a front elevation; the dies are fitted on the bottom A, and the quartz fed through the opening B, whilst the screens are fastened, by nails or screws, to a frame which is firmly secured in grooves provided for its reception at the ends of the mortar, and by two lugs at the bottom of the opening C.

In some cases, instead of employing a double cam, as seen in Plate II. fig. 2, a single one is made use of. This has generally the form shown Fig. 29, and possesses the advantage of allowing the axle to be placed nearer the stamp stem than it can with any other cam, and also that by its use a greater number of blows can be struck per minute, without danger of breakage.

FIG. 29.



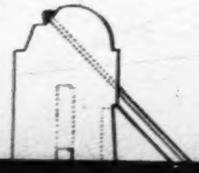
SINGLE CAM.

The auriferous material having been reduced to the state of a finely-divided powder, it becomes necessary to provide means for the concentration and separation of the gold, which is more or less perfectly effected by an almost infinite number of different contrivances, varying slightly in their details in almost every establishment that may be visited. However much the processes employed may differ in this respect, only two decidedly distinct systems are now practically in use in California, viz. amalgamation in the battery; and crushing without the use of mercury, amalgamation being subsequently effected by means of appliances specially designed for that purpose.

Amalgamation in the Battery.—When this method is adopted, the batteries are often provided with amalgamated copper plates *b* (Plate II. fig. 2), about five inches in width, extending the whole length of the box; one on the feed side and the other at the discharge, the latter being protected by the sheet iron lining of the feed hopper, and each having an inclination of from forty to forty-five degrees towards the stampers.

When these are not employed, spaces for the accumulation of amalgam are allowed between the dies and the sides of the box, and vertical iron bars are placed inside the gratings, between which the hard amalgam is found to collect. The copper plates are covered with mercury, by means of a rag dipped in dilute nitric acid, with which quicksilver is rubbed over the surfaces to be coated, in the same way as on those used in ordinary sluices. Quicksilver is also sprinkled into the boxes, by the feeder, at intervals of about an hour, and in quantities varying with the richness of the rock operated on. One ounce of gold requires for its collection about an ounce of mercury; but when the gold is in a finely-divided state, the addition of another quarter of an ounce of this metal is thought advantageous. The proper proportion is, however, readily ascertained by closely watching the discharge. If any particles of amalgam which may pass through be hard and dry, a little more quicksilver must be introduced; but if, on the contrary, they be soft and pasty, or globules of mercury make their appearance, the supply in the battery must be diminished.

When the proportion of mercury has been properly adjusted, the amalgamation of the gold is very completely effected, except in cases in which the ores contain large quantities of lead or antimony, and have been previously burned for the purpose of expelling their more volatile constituents, by which treatment the particles of gold often become coated in such a way as to interfere with their combination with mercury. When the proper proportion of quicksilver has been regularly introduced, and the rock contains coarse gold, from sixty to eighty per cent. of the gold saved is caught in the battery; but when, as in the case of some of the ores of Nevada, the gold is in a very finely-divided state, and is associated with ores of silver and other sulphides, the results obtained are less satisfactory. The alloy resulting from the treatment of such ores contains silver, and in some cases affords from 300 to 400 thousandths only of gold, often producing a spongy amalgam of a dark colour, made up of an aggregation of numerous finely-divided particles. Küstel is of opinion that this effect is partially due to the presence of manganese, but it appears difficult to understand how this substance should influence, to any important extent, the combination of gold and mercury. This amalgam is exceedingly light, and is, therefore, difficult to collect, either by riffles, copper plates, blankets, or any of the other appliances commonly employed for the purpose.



10 STAMP BATTERY

STAMPS 550 LBS.

WET CRUSHING & AMALGAMATING IN BATTERY.

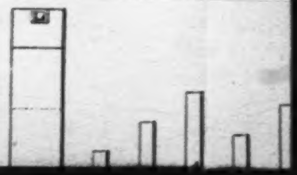
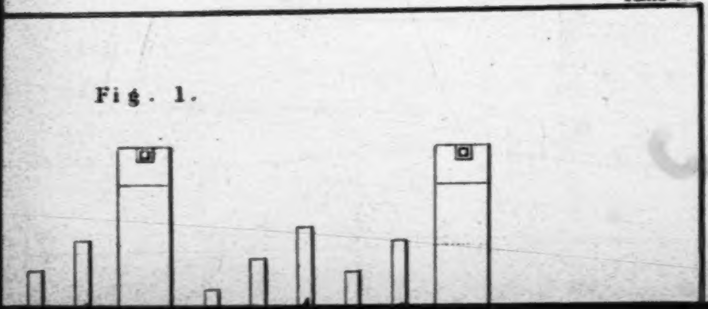


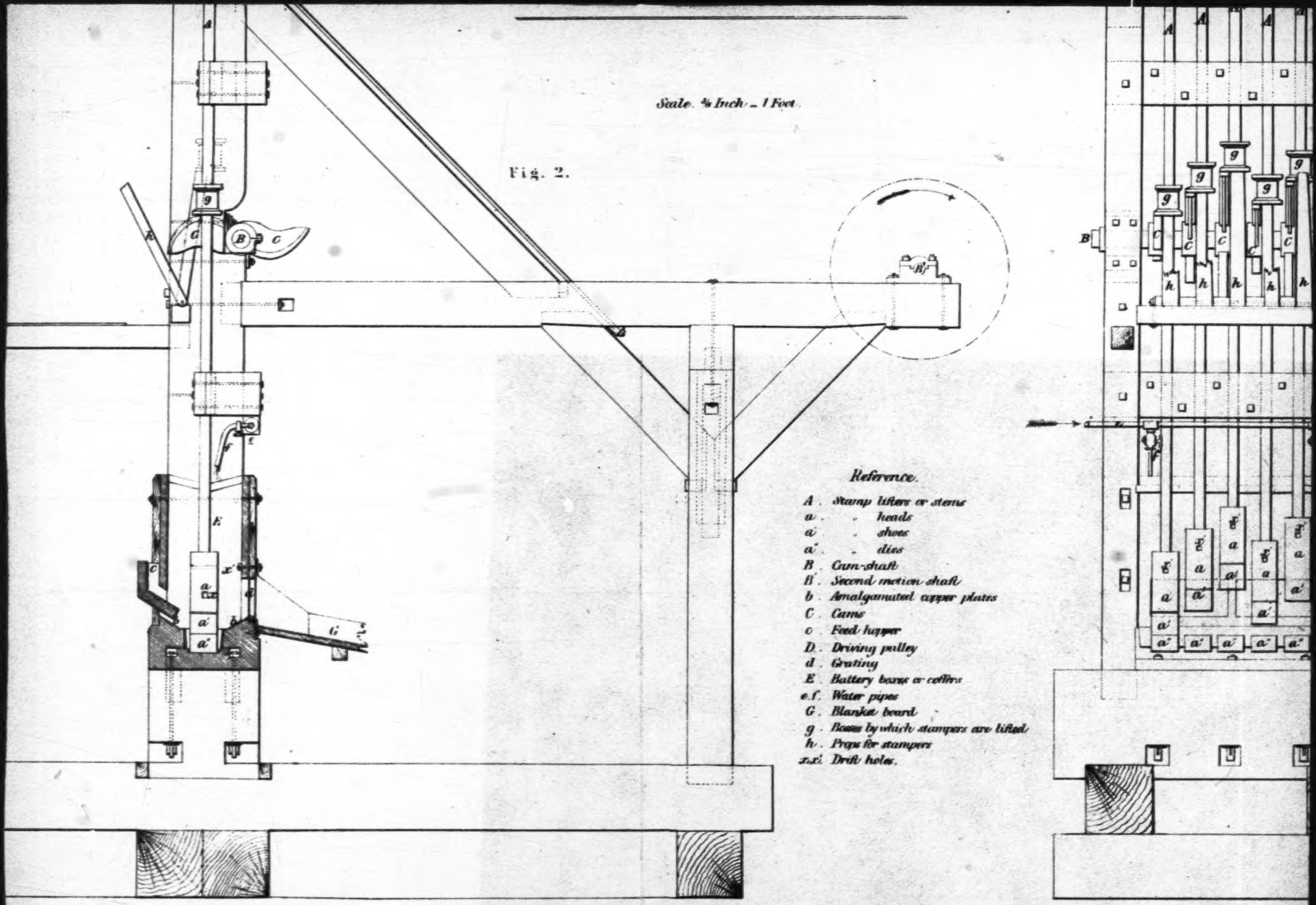
PLATE II.

Fig. 1.



Scale. ¾ Inch = 1 Foot.

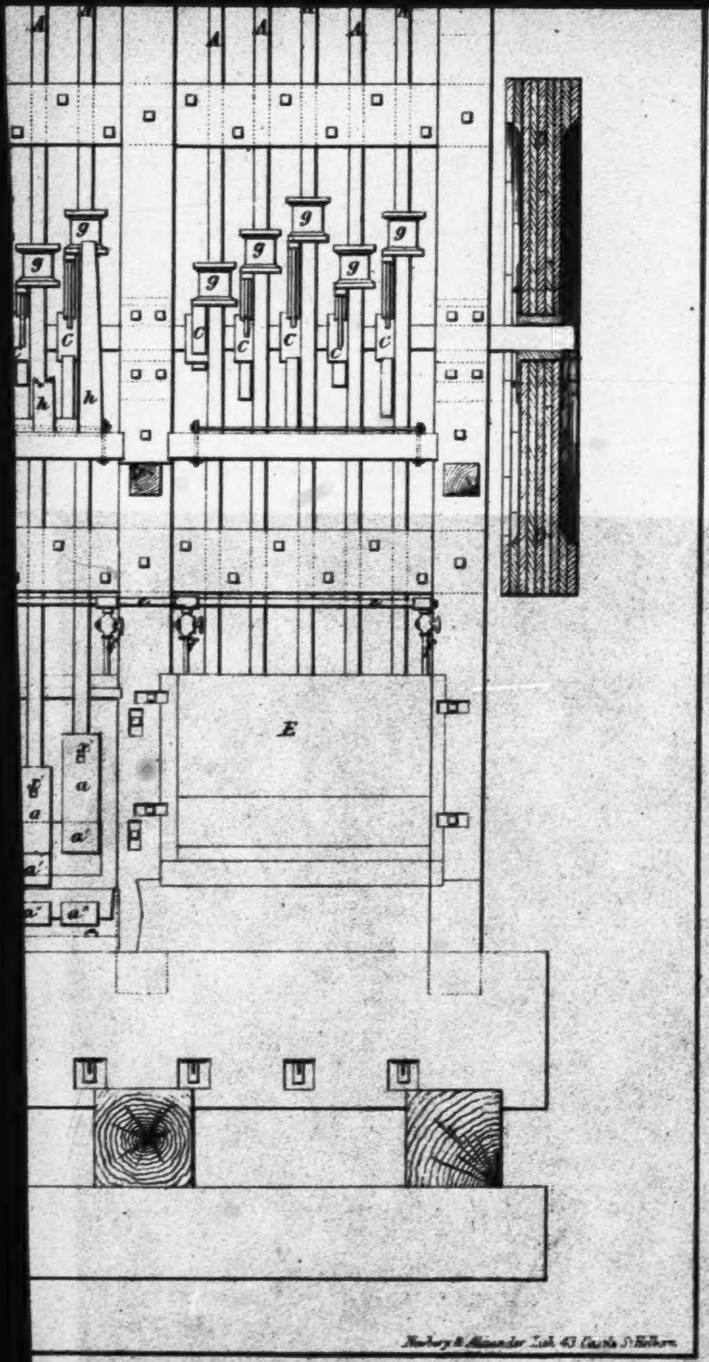
Fig. 2.



Reference.

A . Stamp lifters or stems
a . " heads
a' . " shoes
a'' . " dies
B . Cam-shaft
B' . Second motion shaft
b . Amalgamated copper plates
C . Cams
c . Feed hopper
D . Driving pulley
d . Grating
E . Battery boxes or coffins
e.f. Water pipes
G . Blanket board
g . Bosses by which stampers are lifted.
h . Props for stampers
x.x'. Drift holes.

E. & F. N. Spon. 48. Charing Cross London.



Newbery & Alexander Lith 43 Castle St Holborn

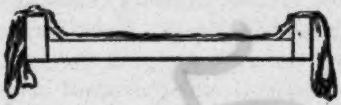
When, therefore, ores contain much lead or antimony, amalgamation in the battery is not to be recommended, since this spongy amalgam is more difficult to retain than the most finely-divided gold, and a large proportion of it floats off over the blankets, riffles, or copper plates, which may be arranged for the purpose of arresting its progress. There is, besides, no evidence that battery amalgamation possesses, under any circumstances, a decided advantage, for gold ores not associated with sulphide of silver, over stamping without the use of mercury, and in some of the most productive gold districts it is seldom resorted to.

In order to collect the particles of gold and amalgam escaping from the battery box, various ingenious contrivances are employed; but as these arrangements differ but little in their details, whether mercury be employed in the battery, or the amalgamation entirely effected after the escape of the pulverised material through the screens, we will proceed to describe the system generally in use in the northern quartz mines, in which the various arrangements are of the most improved description.

Blankets.—At Grass Valley, which, from the richness of its quartz veins and the excellency of the machinery employed for the reduction of the ores, may be considered as the head-quarters of quartz mining in California, the rock is generally crushed without the introduction of quicksilver into the mill. In this district the sand and water issuing from the battery are conducted over blankets spread on the bottoms, and lining the sides of shallow troughs or sluices, inclined at an angle of from three to four degrees with the horizon. Beyond the blankets there are, in most cases, riffles or amalgamated copper plates, which are again followed by some contrivance for collecting the pyrites remaining in the tailings. At the further extremity of this system of appliances there is sometimes a long tail sluice for the purpose of arresting any auriferous material that may have escaped being caught by the other arrangements.

The blanket boards are from fourteen to sixteen inches in width, inside measure, and three inches in depth, being so laid, with a regular longitudinal inclination, that transversely the bottom is perfectly level. It is necessary that this should be carefully attended to, in order that an equal depth of water may, when the mill is in action, flow over every portion of its surface, and thus prevent the occurrence of a rapid current on one side, whilst an accumulation of sand is taking place on the other. The blankets employed are, for the most part, woven expressly for the purpose, of coarse grey wool, and are made of such a width that, when laid in the troughs, and fitting closely over the bottom and sides, they hang down about six or eight inches beyond the top of the woodwork; and in order that they may be readily adjusted, and that no portion of the stream may flow down behind them in the angles, a triangular piece of wood is sometimes fitted into the corners made by the junction of the bottom and sides. The section of one of these troughs (Fig. 30) shows the position of the blanket when fitted in its place.

FIG. 30.



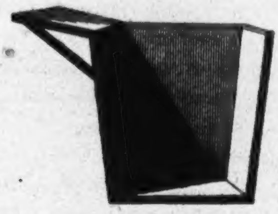
SECTION OF BLANKET BOARD.

In laying these blankets in their troughs, they are so placed as to overlap each other, like tiles on a roof, in order that the water flowing from one may run directly over the other, being prevented by the lap given to them, from finding its way between the blankets and the bottom. The troughs are also themselves made in two or more lengths, and are so arranged that the water in passing through the first, falls into the second, from a height of about three inches.

The general disposition of the batteries and blankets in a well-arranged Californian establishment, will be understood by reference to Plate III., in which fig. 1 is a sectional elevation, and fig. 2, a plan of the apparatus. The battery box A is fitted in front with a water-tight wooden trough B, running its whole length, and provided with an aperture *b*, opening on the blanket board C, and with a second aperture *b*′, at its inner end, which, like the first *b*, can be closed by a small wooden slide, or by placing a bit of plank before it. Immediately before the centre of each battery is placed a blanket board C, and, between each pair a third C′, to be used when either of those on each side of it is thrown out of action, for the purpose of removing the blankets. The troughs C and C′ are sixteen inches in width, three inches in depth, and have a length of twelve feet; these empty themselves into the troughs D D′, having the same width, depth, and inclination, but only nine feet in length. When the batteries are in their normal state of working, the water and crushed ore flow from the apertures *b*, passing through the troughs C, and running over the surface of the blankets with which they are lined; a large portion of the gold and iron pyrites thus becomes entangled in the fibres of the wool, and remains behind, whilst the lighter particles of quartz are carried off by the force of the current, and escape at the lower end of the troughs. If this were allowed to go on without interruption, the fibres of the blanket would soon become so charged with the heavier fragments of crushed ore as at length to cease to act, and the whole of the products of stamping would pass off and be lost. To obviate this, the blankets are frequently washed up, and are then again ready to arrest any particles of gold or pyrites with which they may come in contact. In well-managed mills the blankets at the upper end of the arrangement are now washed every fifteen minutes, and this operation is conducted as follows:—The orifice *b*, opening from the trough B, on one of the blanket boards C, is closed, and the aperture *b*′, communicating with the board C′, standing between the two batteries is opened. This has the effect of leaving the first trough C without water, whilst the stream which before flowed over it is directed through the central trough C′. The blanket over which the current is no longer flowing is now taken up, being at the same time so folded as to prevent the loss of any portion of the adhering matter, and then taken to a rectangular tank prepared for the purpose, where it is carefully washed, and again laid in the trough from whence it was taken. The flow of water and sand from the battery is now cut off from C′, and admitted into its original channel; the same operation being repeated on the trough belonging to the other battery. When the blankets in the intermediate trough require changing, it is done whilst the others are in use. The blankets at the upper end of this system only, are changed so frequently as above mentioned; whilst those on the lower boards D D′, often remain some hours without being washed up. In some mills instead of the intermediate trough C′, there are two blanket boards to each battery, in which case one is being cleaned up whilst the other is in operation.

The cistern, or tank, Fig. 31, in which the blankets are washed, is always situated in the immediate proximity of the troughs containing the blankets requiring most frequent washing, and is generally four feet in height and four feet square at the top, but somewhat smaller at the bottom. On one of its sides is an inclined ledge on which the blankets are rolled after washing, and when again laid down they are rapidly unrolled from the bottom of the trough upwards, so as to fall directly in their places.

FIG. 31.



SECTION OF WASHING TANK.

Sometimes, although rarely, ox hides have been employed instead of blankets, and are placed in the troughs with the grain of the hair against the current, but these are only used in mills of very primitive construction. Sheep skins have also occasionally beeu tried, but these, like ox hides, make bad substitutes for blankets. In some mills, a small stream of clear water is admitted into the troughs at the head of the blankets, for the purpose of rendering the mixture of sand and water issuing from the battery so dilute as to enable the gold and pyrites to settle with greater facility, and this is by some mill-managers considered advantageous; but when the volume of water passing through the troughs is thus increased, their inclination should be made proportionately less considerable. In the majority of cases the gold retained in the battery, together with that collected on the blankets, will represent at least nine-tenths of the total amount obtained from the rock under treatment; but there is, nevertheless, a notable quantity of the precious metal which passes over the blankets, and of which it is desirable to recover the largest possible amount.

Amalgamated Plates.—With this object amalgamated copper plates, often arranged in the form of riffles, are employed, and at the end of these are cisterns or tyes for the collection of the auriferous pyrites, which is subsequently concentrated and treated for the gold it encloses.

At some of the mills in Grass Valley, after running over two blanket boards, respectively twelve and nine feet in length, and having an inclination of three and a half degrees, the water is conducted through troughs E E′ (Plate III. fig. 2) eleven inches wide, of which the bottoms are formed of amalgamated copper riffles, and which have the same inclination as the blankets. From this, the current passes through troughs F F′ set with a less inclination, and of which the bottoms are also formed of amalgamated plates, whilst at the end of these are two reservoirs for the retention of tailings. The plates *e*, forming the bottoms of the riffles in the troughs E E′, are made so as to slide easily in and out of their places for the purpose of being cleaned, or re-amalgamated, and are about eight inches in length.

The riffle plates in the troughs F F′, are also movable, although considerably longer than in those above them; but it is probable that, if instead of contracting these spouts they had been widened, or still better, if two of them had been employed to carry off the current flowing from each of the riffle boxes E E′, and a little fresh water admitted for the purpose of diluting the mixture of water and pulverised ore, the fine particles of gold would have had increased facilities for settling and becoming attached to the surface of the amalgamated plates.

Cleaning up, &c.—The stampers, except when undergoing repair, or stopped for the purpose of cleaning up the gold which accumulates in the battery boxes, are kept constantly at work day and night, the frequency with which the boxes are cleaned of course depending on the richness of the rock operated on. Generally speaking, however, the cleaning up of the battery box takes place at least every week, and often more frequently, particularly when mercury is introduced during the process of crushing. When quicksilver is used in the battery, a very large proportion of the gold obtained is taken from it in the form of amalgam, and even when this metal is not introduced the cleaning up of the battery affords a considerable percentage of the produce, which accumulates in the cavities around the dies, in the form of metallic spangles. The coarser the gold in the rock treated, the larger will necessarily be the percentage of the total produce retained in the battery. Instead of blankets, employed as above described, Brunton's separators, with revolving woollen cloths, have been sometimes used; but these not having been found efficient, have been generally abandoned.

Settling Pits.—When settling pits are used for the purpose of collecting the tailings for subsequent treatment, it is necessary that at least two of them should be provided, so that whilst one is being filled the other may be cleaned up, but in many cases a long pit resembling a Cornish tye is substituted, which, instead of being allowed to fill, and then emptied, is constantly under the superintendence of a lad, who shovels the sand out of it as rapidly as it is brought in by the current, thus causing an amount of agitation, resulting in a further portion of the lighter particles flowing off, and conducing to a greater degree of concentration in the product obtained. It is, moreover, necessary that every mill should be furnished with at least two tanks for washing up blankets, so that one may be ready for use whilst the other is in course of being cleaned up. In some mills, also, the tailings, instead of being allowed to accumulate in a reservoir or tye, prepared for that purpose, and from which they are subsequently removed by manual labour before being submitted to further treatment, are conducted directly into concentrating apparatus, from the end of troughs lined with amalgamated plates.

Amalgamator.—The separation of gold from the matters caught on the blankets, and collected in the washing tanks, is generally effected in California by a very simple piece of apparatus, introduced, many years since, into the northern mines by Mr. M. Attwood, Figs. 32, 33.

This machine consists of two wooden rollers A, eight inches in diameter and two feet in length, furnished on their circumference with numerous small flat knife-bladed pieces of iron arranged around them with their edges at right angles to the axes of the cylinders, and working in cisterns containing mercury; above these rollers, which are set in motion by the pulleys B, over which a belt is stretched so as to cause them both to revolve in the same direction, but contrary to that of the water flowing through the apparatus, is a hopper C, for receiving the sand to be washed. Another pulley D, is connected by a second belt to a rigger keyed on a small shaft fixed in the roof of the mill house, and communicating motion to the whole arrangement. Below the cylinders A, is a riffle board E, having an inclination of seven degrees from the horizontal, and generally covered with plates of amalgamated copper, which can be readily slipped out for the purpose of having the gold amalgam, which may become attached to them, removed. When copper plates are not employed for this purpose, the steps of the riffle are reversed and charged with mercury.

To use this apparatus, some of the sand taken from the cistern in which the blankets have been washed, is placed in the hopper, and a small stream of slightly-warm water allowed to play on it from C′, in such a way as to gradually wash it under the spiked cylinders A, and from thence over the amalgamated riffle board E.* This riffle board is

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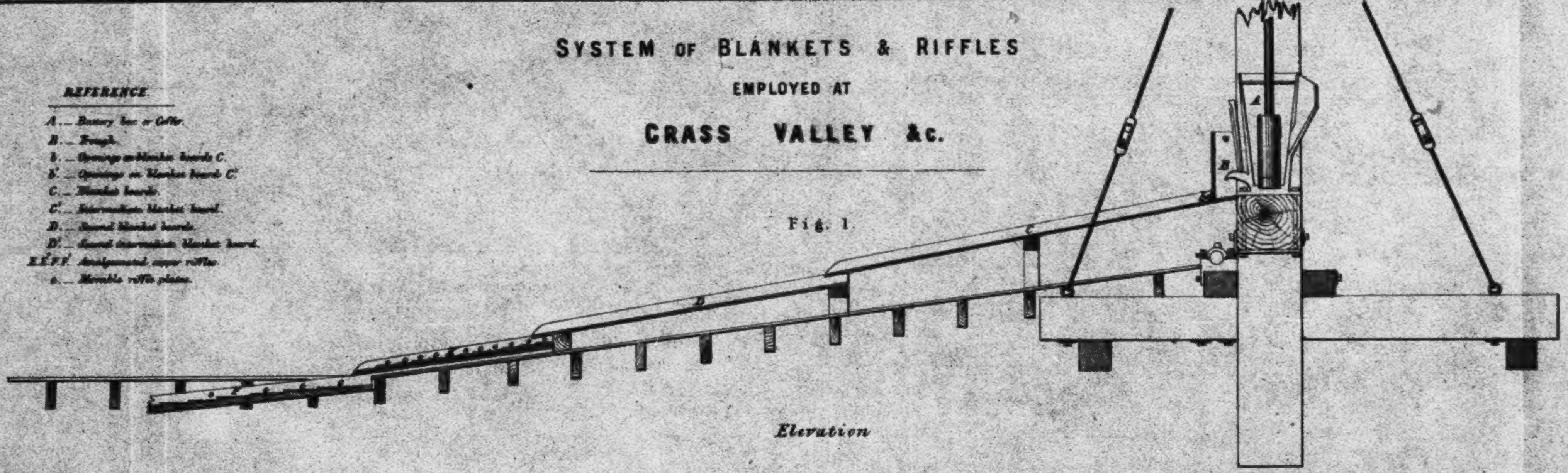
* The use of warm water is found to facilitate amalgamation, particularly during cold weather.

SYSTEM OF BLANKETS & RIFFLES
EMPLOYED AT
GRASS VALLEY &c.

REFERENCE.

A. — Battery box or Coffer.
B. — Trough.
b. — Openings in blanket boards C.
b'. — Openings in blanket board C'.
C. — Blanket boards.
C'. — Intermediate blanket board.
D. — Second blanket boards.
D'. — Second intermediate blanket board.
E.E'.F.F'. Amalgamated copper riffles.
e. — Movable riffle plates.

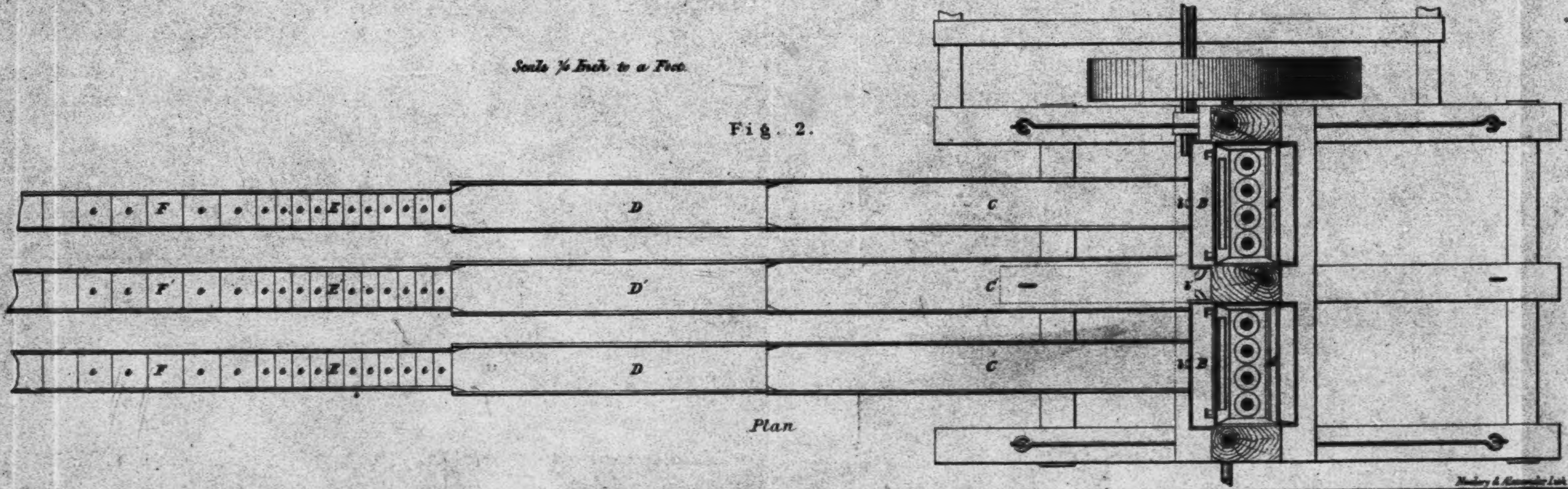
Fig. 1.



Elevation

Scale ½ Inch to a Foot.

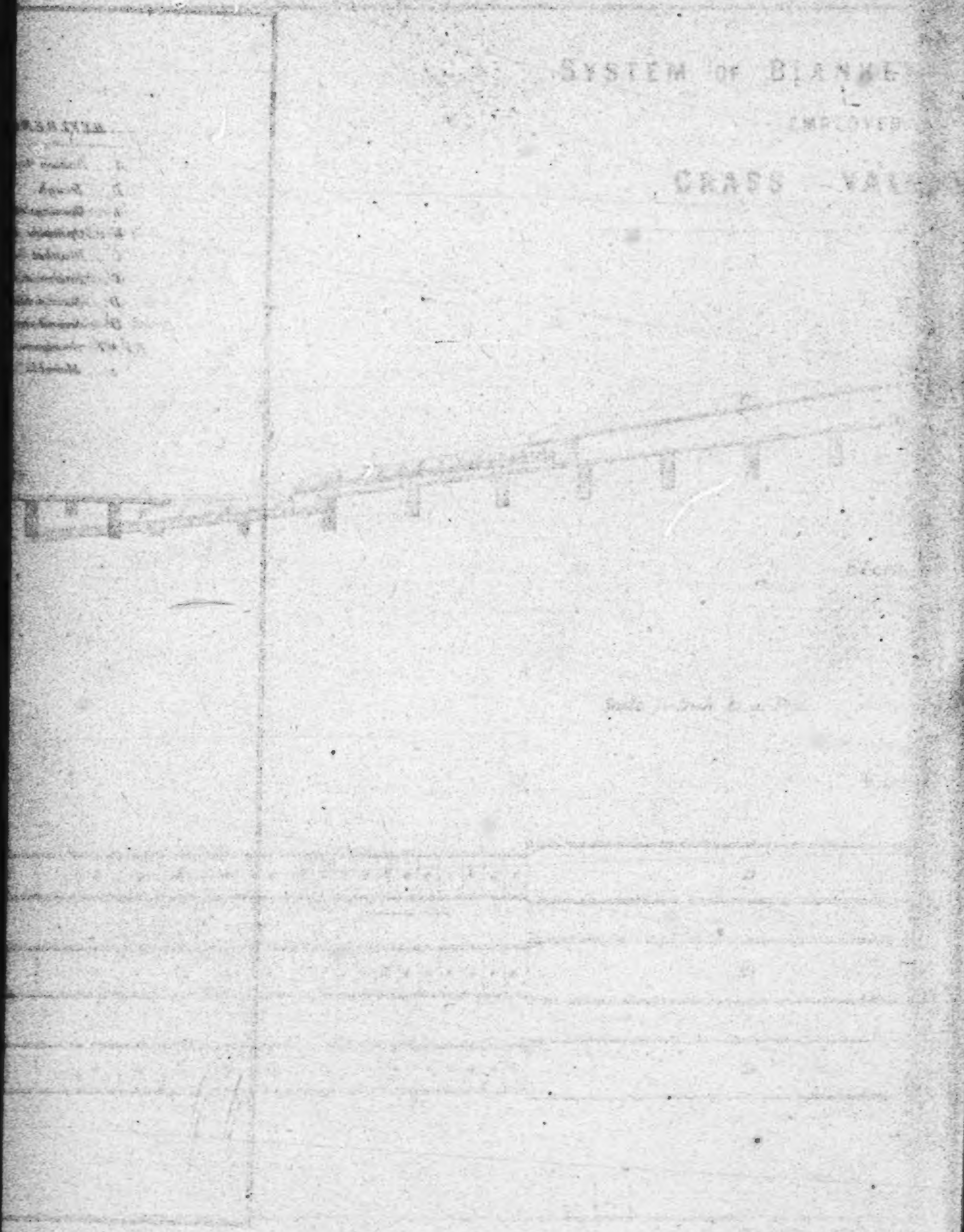
Fig. 2.



Plan

Newbery & Alexander Lith. 43 Castle St Holborn.

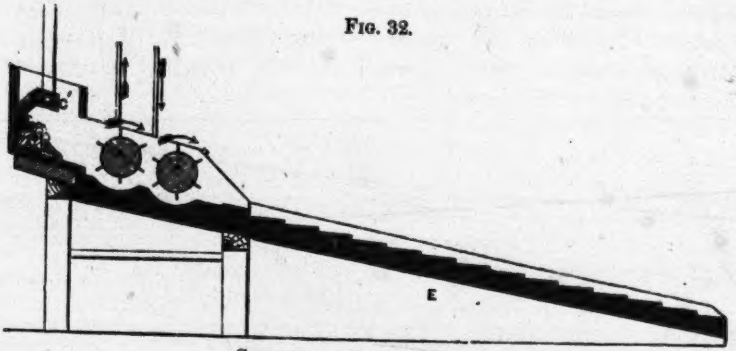
E. & F. N. Spon. 48, Charing Cross, London.


SYSTEM OF DRAINA
IMPROVED
GRASS VAL

usually nine feet in length, is divided into several channels, and has at its end a cistern for retaining the pyrites and other matters, which, not combining with mercury, escape amalgamation.

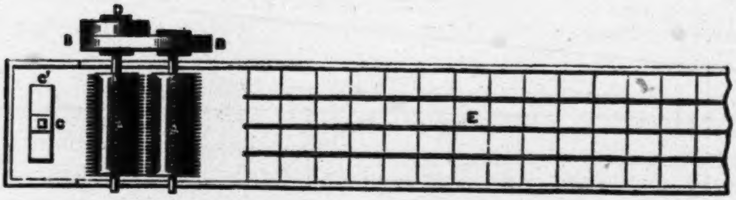
As the material operated on in this machine is always highly auriferous, and consequently very valuable, great care is taken to exactly regulate the feed at the upper end, and to keep the surface of the riffles perfectly bright, and clear of any accumulation of pyrites. A boy is therefore stationed at the riffle board, who skims off any scum or impurity which may accumulate on the surface of the plates, and collects it for subsequent treatment in small cast iron arrastres. The sulphides deposited in the tank at the extremity of the riffle board are sometimes ground with mercury, in a Varney or Wheeler pan, and, after thus extracting as much gold as can be so obtained, they may, if they

FIG. 32.



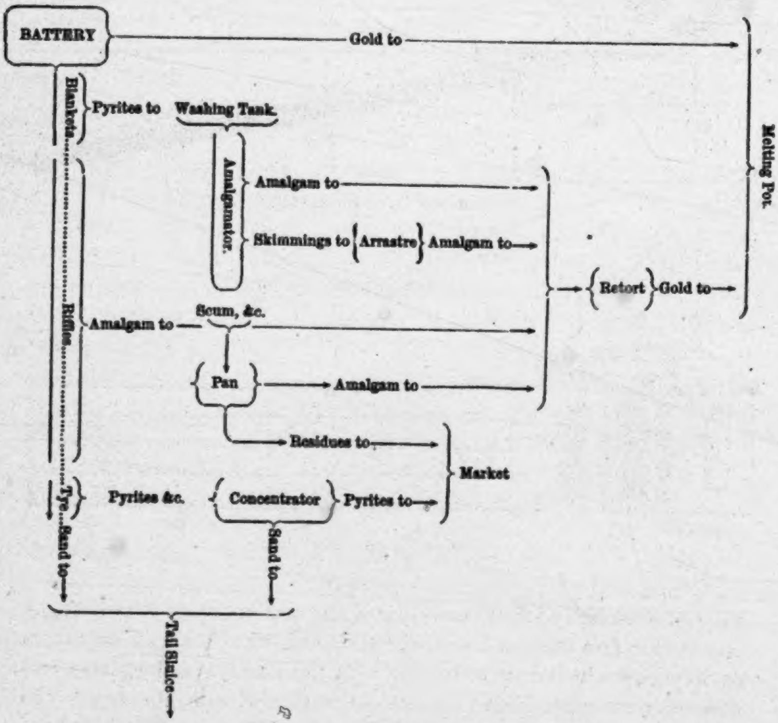
SECTION OF AMALGAMATOR.

FIG. 33.



PLAN OF AMALGAMATOR.

still retain a sufficient amount of gold, be drawn off, and, after settling, are collected and sold for treatment by smelting or chlorination. In many establishments, however, the pyrites from the tanks at the end of the amalgamators, undergoes no further treatment, but is collected and sold as an auriferous sulphide of iron, or it may, previous to being sent to market, be subjected to a simple washing in a rocker, or otherwise, for the purpose of eliminating the sand. The annexed diagram shows the various processes to which gold-bearing quartz is subjected in the neighbourhood of Grass Valley, and explains at a glance the series of operations which the rock undergoes in some of the best mills in that district.



It must, however, be understood, that there are scarcely two mills in which the various manipulations are conducted in precisely the same way, and consequently in the chart of operations mere non-essential differences have been disregarded.

Attwood's System.—Mr. M. Attwood, of San Francisco, has recently designed an arrangement, of which a drawing is given, Plate IV., for saving the gold issuing from a stamping mill. In this Plate, fig. 1 is a sectional elevation; fig. 2, plan; fig. 3, lower end of tyes; and, fig. 4, end of steam chest.

In this arrangement Mr. Attwood does not make use of blankets, but the ground ore issuing from the battery screens, flows directly on to the amalgamator, where it is gently stirred by the action of the cylinders A, turning in the direction indicated by the arrows, and then passes on to a riffle board B, covered by amalgamated copper plates, where a great portion of the amalgam, escaping from the cast iron mercury boxes *a*, will be collected. In order that the mercury in the boxes under the rollers may not become too cold, and its affinity for gold be thus rendered sluggish, they are cast with a double bottom, through which a current of steam can be made to pass, and which is easily regulated by an ordinary tap.

From the riffles B, the ground material passes into the tye C, of which the bottom is inclined at a considerable angle, and which is provided at the lower end with a slot *c*, for regulating the depth of water within it. This is done by means of the stops *c*′. In order to catch any globules of soft amalgam or mercury, which may become detached from the surface of the amalgamated plates, a small cistern D, running the whole width of the riffle board, is provided; in this is an agitator *d*, turning in the direction indicated by the arrow, and which constantly keeps the box, to the depth of its arms, free from accumulations, so as to form a depression in which the mercury and amalgam may become deposited.

To use this apparatus, one of the stops *c*′ is placed in the slot *c*, and the mill started in the usual way; the sand which has passed through the amalgamator soon reaches the tye, and the heavier portions begin to accumulate behind the stop, whilst the lighter particles are carried off by the current. The removal of the light sand is facilitated by gently sweeping the surface of the deposit upwards against the stream with a light broom, a boy being stationed there for that purpose; and when the pyrites which is deposited, accumulates to the height of the top of the first stop, another is inserted, and the operation carried on continuously. When one of the tyes has been filled in this way, the tongue E is so turned as to direct the sand and water into the other, which is thus filled whilst the first is being emptied.

It is evident that by this means the pyrites will be collected in the tyes in a very concentrated form, and that the amount of labour required is but small; we have, however, never seen this apparatus in operation, and are without any precise data showing its efficiency, as compared with the blankets and riffles now in general use.

System employed in Australia—Port Phillip Company.—Mr. Bland describes the treatment to which the auriferous quartz is subjected, as follows:*—

"The quartz when brought to the surface is separated into two lots. The small size is tipped into a large hopper, and from thence drawn and delivered, direct, to the stamps, and the larger lumps are sent to the stone-breaking machines to be reduced to a size suitable for the stamps. Two of these machines are in constant use, working on an average about ten hours per day. Each machine will break about eight tons per hour when in good order, at a cost, including wear and tear, of about 10*d.* per ton. They are driven by a shaft from one of the battery engines, and take about twelve horse-power to drive the two. The number of stamps at work is eighty, as follows:—Fifty-six heads of about six cwt. each, including lifter and tongues, driven by one twenty-four-inch engine, giving seventy-five blows per minute, taking about one horse power per head, and crushing an average of about two tons four cwt. per head, for twenty-four hours; twenty-four heads of about eight cwt. each, including lifter, &c., driven by a twenty-four-inch engine, giving seventy-five blows per minute, taking in the aggregate about thirty horse power. These stamps crush about four tons per head per diem, and they have a larger proportion of the small material sent to them. The average quantity of quartz crushed per week, of five days, is about 1,130 tons.

"The following is a return of quartz crushed for the twelve months ending September 1865:—

54,413 tons	20,596 oz. 10 dwt. 12 gr.	7 dwt. 13 gr.

"The gold above mentioned was collected in the following way:—

In stamp beds	66·08 p. c.
Mercury boxes	22·95 p. c.
Blankets	10·97 p. c.

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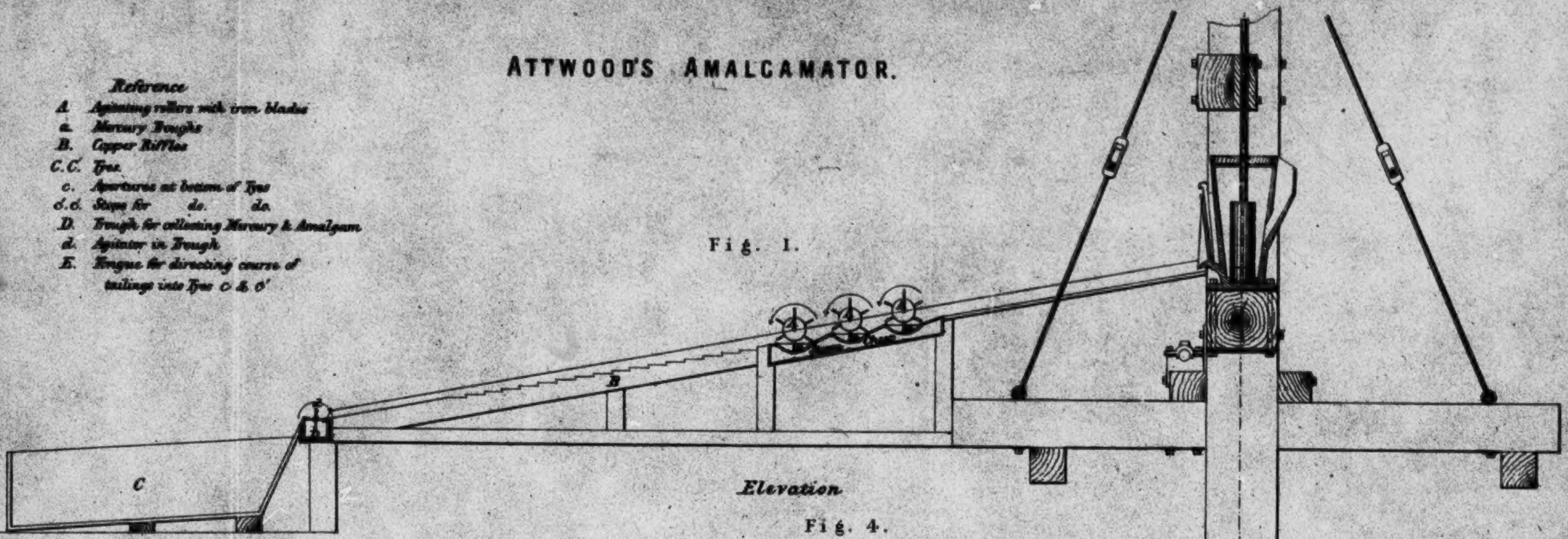
* The quartz, at the works of the Port Phillip Company, is stamped much coarser than in California. The screens have only ten holes to the lineal inch, and hence the larger amount got through within a given time.

ATTWOOD'S AMALGAMATOR.

Reference

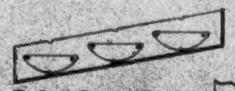
A.	*Agitating rollers with iron blades*
a.	*Mercury Troughs*
B.	*Copper Riffles*
C.C.	*Tyes.*
c.	*Apertures at bottom of Tyes*
c.c.	*Stops for do. do.*
D.	*Trough for collecting Mercury & Amalgam*
d.	*Agitator in Trough*
E.	*Tongue for directing course of tailings into Tyes C & C'*

Fig. 1.



Elevation

Fig. 4.



End of Steam Chest

Fig. 3.

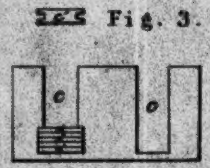
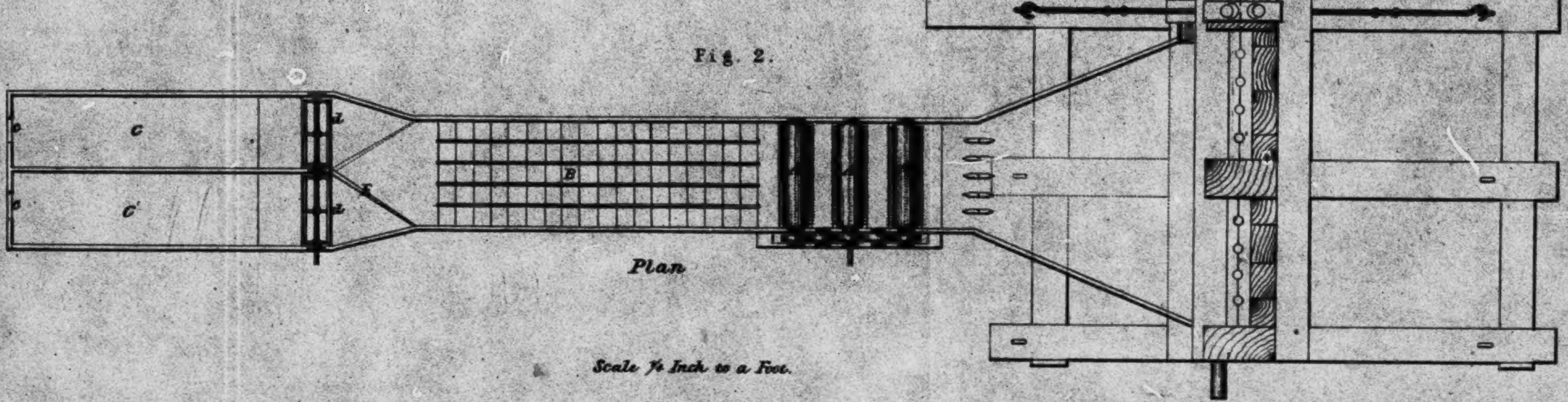


Fig. 2.



Plan

Scale ¾ Inch to a Foot.

Newbery & [illegible], Lith. 43, Castle S^t Holborn.

E. & F. N. Spon, 48, Charing Cross, London.

"The quantity of water required to work the stamps efficiently is about eight gallons per head per minute, which is 921,600 gallons per diem. The tailings on leaving the stamps run into settling boxes, where the current is checked, and the heavier material settles. These boxes are cleaned out every few hours, and the material sent to the buddle, where it undergoes a further concentration, and is dressed up to an average of three or four ounces of gold to the ton of material. This is then sent to the roasting furnace, and afterwards ground in Chilian mills with mercury, and an average of about eighty-five per cent. of the assay contents of the gold is thus extracted. The cost of operating on the pyrites, including the buddle, roasting, grinding, loss of mercury, &c., averages about 2*l.* 14*s.* per ton, or about 1*l.* per ounce of gold obtained.

"The expense will diminish as the quantity of pyrites increases, and improves in richness.

"The quantity of quartz crushed, and yield of gold from the commencement of the Company's operations in 1857 to the 31st July, 1866, was as follows:—

Quartz crushed.	Yield of Gold.
308,661 tons.	180,723 oz. 15 dwt. 10 grs.

Equal to six tons of 2,000 lbs. each."

Loss of Gold, &c.—In California the tailings escaping from the last of the appliances employed for the separation of gold, are never carefully and regularly assayed, and consequently it would be impossible to arrive at a correct estimate of the losses resulting from the imperfections incident to the modes of treatment adopted. It would, however, seem that there is comparatively little difficulty in effecting the separation and amalgamation of the free gold, and that the principal losses attending the working of ordinary gold quartz, arise from the escape of small particles of this metal enclosed in pyrites, and which the concentrating apparatus fails to collect. Nevertheless, in some instances the surface of the gold would appear to be coated with a thin glaze of silica, or some other substance which protects it from amalgamation, and in such cases its combination with mercury can only be effected by grinding in a pan, arrastre, or some similar contrivance. The only data relative to this subject which we have been able to procure, have been obtained from the officers of the Port Phillip and Colonial Gold Mining Company, at whose establishments, in Australia, the tailings are regularly sampled and assayed, and found (see table, page 116) to contain on an average about 2 dwt. of gold, of which some portion is again recovered from the washed sulphides, by roasting and grinding in Chilian mills.

The weight of the stampers employed in the Californian mills differs considerably, and, as a natural consequence, the amount of work done by each in a given time varies in nearly the same ratio. In the neighbourhood of Grass Valley, heavy heads are almost universally employed. At the North Star works, the stampers, including the stems or lifters, weigh 9 cwt. and crush weekly 13¾ tons of quartz, or very nearly two tons each in twenty-four hours. The stampers at Allison Ranch are of the same weight, and perform the same amount of work. At the Eureka mill the stampers weigh 840 lbs. and are estimated to crush about two tons each in the course of the twenty-four hours. The table at the end of this chapter, constructed from data collected by Mr. Ashburner in 1861, gives the cost of stamping, weight of stampers, loss of mercury, and many other particulars relating to the various quartz mills then at work in California, since which period the cost of treatment has been somewhat reduced.*

Concentration of Tailings.—In the earlier days of quartz mining, the pyrites and other metallic sulphides were generally allowed to escape with the earthy and silicious portions of the vein-stuff, and a considerable loss was naturally the result, although an attempt to extract the gold which they enclose was sometimes made by roasting the quartz in kilns or heaps, before subjecting it to the action of the stamping mill. This was, however, found to be of little practical advantage, and quartz is therefore now stamped without any kind of preparation beyond that of *spalling*, or breaking it into fragments of a convenient size, but much care is devoted to recovering from the tailings the largest practicable proportion of auriferous material.

The machinery and contrivances employed with this object are of the most varied description, and it would be therefore impossible to attempt a description of the whole of them, and we shall consequently confine ourselves to noticing such as are most commonly made use of in the best-conducted establishments.

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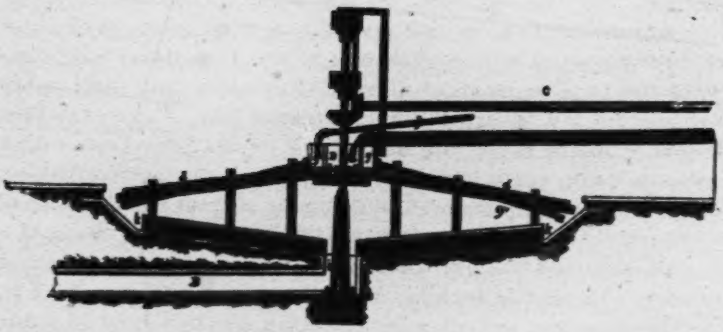
* An interesting paper has been recently published by the Geological Survey of California, entitled "Mining Statistics, No. 1. Tabular Statement of the Condition of the Auriferous Quartz Mines and Mills in that part of Mariposa and Tuolumne Counties, lying between the Merced and Stanislaus Rivers, by A. Rémond."—Philadelphia, 1866.

From this it would appear that the cost of milling at the present time is lower than when Mr. Ashburner compiled his tables; but the information afforded being of a comparatively local character, we abstain from making extracts.

Rocker.—The rocker consists of a trough about twelve feet in length, fourteen inches in width, and ten or twelve inches deep at the sides. In appearance this contrivance is not unlike a school form, of which the top constitutes the bottom of the trough, and of which one leg is shorter than the other, so as to give a slight inclination to the machine. The two legs, which are very short, are rounded at bottom, like rockers, and kept in their positions by articulations which allow of the arrangement working from side to side, this motion being communicated to it by means of a crank, or eccentric and small sweep rod, attached to a pulley, driven by a narrow belt. The trough thus formed is rocked at the rate of forty-five strokes per minute, with a one-inch throw, and is furnished with an end at the higher extremity only, the other remaining open, and forming a spout for the escape of water and sand. Water is admitted at the upper end by means of a flexible tube, and the bottom is lined with sheet iron to prevent wearing. To use this machine, the workman stands at one side of it, near its upper end, and after having turned on the supply of water, he throws in a few shovelfuls of the sand, which has been collected either in the settling pits or tyes. The rocking motion, together with the stream of water, aided by the judicious use of the shovel, causes the lighter silicious particles to be carried off, whilst the heavier pyrites remains in the trough of the rocker. This is removed by the shovel, and another charge of unwashed tailings introduced, the operation being carried on continuously in the same way. This machine washes the pyrites very clean, but appears to allow too large a proportion of it to be carried off in the final tailings.

Concave Buddle.—This apparatus, invented by Mr. Hundt, a Prussian engineer, and patented in this country by a Mr. Borlase, has been successfully introduced at the Port Phillip Company's works in Victoria. This arrangement possesses the advantage of affording a large working area at the head, and at the same time effects a better separation of the waste than can be produced by round buddles of the ordinary construction. Also, when the lighter portions of the tailings have become separated from the heavier near the periphery of the circle, the area over which they are distributed gradually diminishes, which by increasing the rapidity of the flow, enables them to be more readily and effectually carried off. In Australia this apparatus is employed for concentrating tailings from which a large proportion of the gold has been previously extracted by the usual appliances. The woodcuts, Figs. 34, 35, represent a plan and section of one of the best forms of the concave buddle.

Fig. 34.



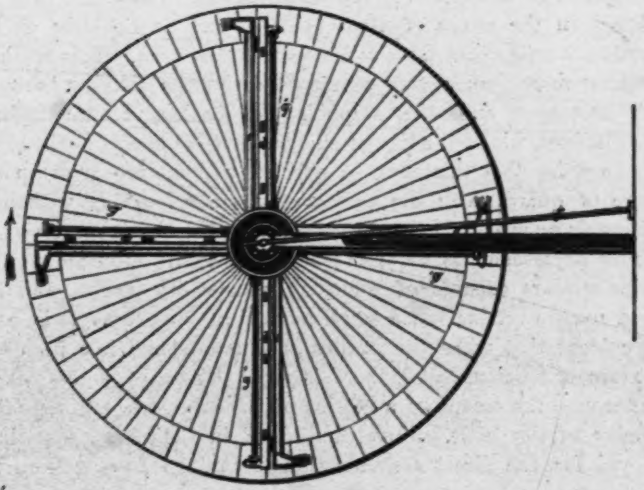
Section of Concave Buddle.

The spout bringing in the mixture of water and sand is represented by A; B is the outlet for carrying off the earthy impurities, or final tailings; C, shaft communicating motion to the buddle arms *d*, the distributing launders *e*, and pipes *g'* attached thereto; *f*, pipe for supplying clean water to the annular cistern *g*, from whence it passes by the pipes *g'* with rose apertures at the ends, and serves to dilute the mixture of water and tailings discharged by the launders *e*, on the annular incline at the periphery. The whole of this arrangement revolves on the shaft D; *i* is a circular pit, into which the final tailings fall, previous to being carried off by the channel B.

To the wooden bars *k*, are attached pieces of canvas which sweep over the surface of the stuff deposited in the buddle, and keep it even and free from ruts. The tailings, entering the receiver *h*, are distributed at the periphery of the buddle through the four launders *e*, which at their extremities are turned at right angles to the direction of their motion, when in action; and at the same time clean water is distributed by the apertures pierced in the terminations of the pipes *g'*.

The speed given to this arrangement of arms, launders, and pipes, revolving on the shaft D, varies in proportion to the state of division of the sands to be treated; when these are rather coarse, the machine may make from six to eight revolutions per minute, but when very fine stuff has to be dealt with, the speed is considerably increased. The influx of tailings and water must be regulated in accordance with the speed of the arms and the density of the stuff operated on; and although no very definite instructions can be given with regard to this subject, a short trial of the apparatus will enable any intelligent workman to make the necessary adjustments. The bed may be from twelve to eighteen feet in diameter, and have an inclination of from six to nine inches from the edge to the centre.

FIG. 35.



PLAN OF CONCAVE BUDDLE.

At the Port Phillip Works, the tailings cleaned by machines of this description are subsequently roasted, and passed through Chilian mills; but where, as in California, the enriched pyrites has to be transported to considerable distances, it would require to be more than once passed through the machine, or, after being once buddled, the heads might be further enriched, either in the rocker, or by a hand buddle or shaking table. A buddle of this kind can be filled in about four hours, and forms an excellent apparatus for enriching ores with but little waste. If it be intended to dress sulphides directly from the riffles, so as to render them almost entirely free from silicious matter, the first heads will require to be re-washed at least once, and the second heads twice; but when this is done, it is necessary to be provided with other buddles besides those which first receive the tailings direct from the riffles, and which will be constantly in use for that purpose.

When the tailings to be washed are not conducted directly from the blanket boards, but are taken either from tyes or the heads of other buddles, they are charged with a shovel into a hopper connected with a circular sieve, working in water, which discharges into the spout A.

Bradford's Separator.—Various modifications of this machine are employed in the mines of Grass Valley, and particularly at the Norambagua mill, about three miles south of the town. This machine is nothing more than a very compact form of the shaking table, by which the pyrites and other sulphides are discharged over the head of the platform, whilst the lighter silicious matters escape at the lower end. In order that this may be effected without too great an admixture of quartz with the pyrites on the one hand, or without a considerable quantity of pyrites escaping with the quartz on the other, very nice adjustment is required; but, if carefully attended to, this machine appears capable of affording satisfactory results, and the washed pyrites retains but a small amount of impurity. It is, however, evident that when the pyrites is thus rendered free from any admixture of silicious sand, there is some danger of a loss taking place through the escape of a portion of the sulphides from the other extremity of the table. These tables are, in California, sometimes employed for the direct concentration of tailings flowing from the riffles.

The tailings enter by a spout, five inches wide, but which expands at the extremity into a fan-shaped form, so as to be of nearly the same width as the table itself. This has its end closed by a fillet, in which is a line of numerous small holes, fitted with regulating pegs. The table is formed of a perfectly flat sheet of copper, two feet two inches in width, and about three feet in length, suspended by iron rods in such a way that its degree of inclination can be readily varied by means of simple regulating gear. In order to render the motion perfectly regular, the table is held in its position by means of spiral springs, attached near the middle of each side, and connected with a wooden framing. The necessary shaking motion is imparted to the table by rods on either side, moved by eccentrics on a shaft, on which is a small fly wheel, which is driven by a pulley and belt connected with a conical roller, opposite to which is a reversed roller of the same kind, with which it is connected by a belt, by changing the position of which, the speed may be easily regulated. The throw of this table is about two inches, and the speed at which it works very great. When this machine is in operation, the pyrites is discharged over a lip into a small box, whilst the impoverished tailings pass off at the other extremity, and are washed away by a stream of water. This is a very ingenious apparatus, and may be made to do its work remarkably well; but the rocker is, nevertheless, more frequently employed, and possesses the advantages of being cheaper, doing its work more expeditiously, and not requiring such nice adjustment.

In addition to the contrivances which have been described, innumerable appliances have been invented, and in many cases patented, for the reduction and amalgamation of gold quartz, and the concentration of tailings. An immense amount of ingenuity has been expended on various machines for effecting all these operations at a cheap rate, and some of them bear evidence of great mechanical skill: up to the present time, however, it may be certainly said that most of them have been more extensively advertised than employed.

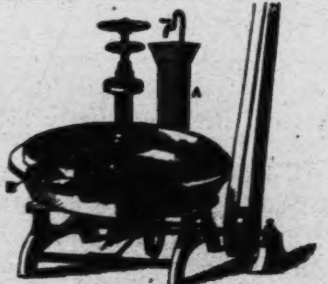
Extraction of Gold from Sulphides.—The Hepburn and Peterson, as well as the Wheeler, Varney, and other pans, extensively employed in Nevada for the treatment of silver ores, are sometimes used for this purpose; but as these machines were originally designed for the extraction of silver from its ores, they will be described under the head of silver.

The arrastre, used in the Grass Valley district for the treatment of scrapings taken from the copper plates of the amalgamator, and other residues rich in gold, consists of a basin of cast iron, four feet in diameter, and of which the mullers are also generally of iron, although they are sometimes, but more rarely, made of stone. These are set in motion by a central shaft connected by a belt with the other machinery of the mill, the cleaning up being effected in the usual way, except that, as the iron pan is free from joints in which an accumulation of mercury and amalgam could take place, it is more easily managed than the same machine with a stone bottom.

In some establishments, as at Allison Ranch, the tailings are passed through a series of iron basins warmed by steam, and provided with mullers, not unlike, in their construction, those of the ordinary Wheeler pan; but they are less efficient than the pans employed for the treatment of silver ores, and are by no means generally adopted.

Baux and Guiod's Amalgamator.—This machine, which is represented Fig. 36, is occasionally used for the extraction of gold from tailings.

Fig. 36.



Baux and Guiod's Amalgamator.

In this apparatus the water and ores are introduced at the bottom of the pan, through a hopper bolted on its side, and the discharge of tailings is so arranged as to be from the highest point, which is the centre of the lid, whilst the substances being ground are thrown by centrifugal action upon a bed of mercury lying in a groove around the internal circumference. This apparatus is worked continuously, the tailings from the blankets entering at A, and flowing off by the spout B, cast on the cover of the machine.

Chlorination Process.—Several establishments employing this process are carried on, on a small scale, in the neighbourhood of Grass Valley; and when the gold is in a finely-divided state, satisfactory results are obtained.

The concentrated tailings are first roasted in reverberatory furnaces of the ordinary construction, generally heated by wood, until no further smell of sulphur is evolved, a little charcoal or salt being sometimes added, towards the close of the operation, for the purpose of decomposing any sulphates or arsenical salts which may have been formed. This roasting is conducted at a very low temperature, and consequently ordinary bricks are alone employed in the construction of the furnaces, which are of sufficient capacity to work a charge of from a ton and a half to two tons. After the expiration of from six to eight hours the charge is withdrawn and spread evenly on the floor to cool, after which it is repeatedly sprinkled with water, and turned over in order to get it regularly and suitably moistened throughout, since on the degree of humidity of the mass greatly depends the success of the subsequent operations. When properly moistened, the roasted pyrites is introduced into large wooden tubs, about seven feet in diameter, and from twenty-five to thirty inches in depth. These tubs are provided with false bottoms, beneath which chlorine gas is introduced, and allowed to permeate the mass of damp auriferous oxide of iron. At the bottom of each tub are two holes, one for the introduction of chlorine through a lead pipe connected with a leaden gas generator, and the other for running off the solutions. The gas is produced from a mixture of common salt, peroxide of manganese, and sulphuric acid; and after covering the tub, and keeping up the evolution of gas during from twelve to fifteen hours, the cover is removed and clean water introduced. Water is thus added until it reaches the surface of the charge, when the discharge pipe is opened and the liquid, containing chloride of gold in solution, drawn off into glass carboys. Solution of sulphate of iron is now added, which precipitates the gold in the form of a dark brown powder, readily separated by decantation, and filtration, and subsequently melted into bars, which generally contain about 995 thousandths of gold.

This process, when the gold is in a finely-divided state, affords good results; but the larger particles of metal not being dissolved in the time necessary for effecting the solution of the smaller ones, they are often partially attacked only, and unless great caution be exercised, a loss is the result. A sample of the residues from the chlorination vats of one of the establishments at Grass Valley, afforded us, by assay, 18 dwt. of fine gold per ton.

The compositions of three samples of pyrites concentrated from tailings, are given in the following analyses by Mr. F. Claudet, who has kindly communicated his results.

ANALYSES OF AURIFEROUS CALIFORNIAN PYRITES, CONCENTRATED FROM TAILINGS.

	From Grass Valley.	From near Sonora.	North Star, Grass Valley.
Sulphur . . .	46·700	37·250	43·720
Arsenic	0·310	8·490	1·360
Iron	41·650	36·540	39·250
Copper	trace	trace	0·220
Lead	„	0·400	trace
Gold	0·037	0·302	0·026
Silver	0·036	not determined	0·012
Cobalt	„	„	0·150
Silica	10·970	17·180	14·230
	99·703	100·162	98·968
	Per Ton of 20 cwt.		
	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.
Gold	12 2 0	98 13 0	8 10 0
Silver	11 16 0	...	3 18 0

Amalgamation in Pans.—Instead of stamping auriferous quartz, either with or without mercury, in the battery, and subjecting the sands flowing from the gratings, to the different operations which have been described, they are sometimes, on issuing from under the stampers, at once collected in settling pits, from whence they are removed by manual labour, in order to be afterwards ground in pans, similar to those employed for working silver ores, and which will be described when treating of that subject. However suitable this method may be for operating on small quantities of very rich quartz, it is evidently not adapted for working large amounts of moderately auriferous rock, and we are not aware of there being, at the present time, any establishment operating extensively and regularly on this principle.

Retorting.—The amalgam of gold and mercury collected during the progress of the various operations connected with the treatment of gold quartz, is first filtered, in order to remove the excess of mercury, and afterwards retorted and melted into bars for the market. In California this expulsion of the redundant mercury is generally effected by well wringing the amalgam in a buckskin, although a piece of closely-woven canvas may, after being wetted, be employed for the same purpose. The amalgam is in this way formed into balls of about the size of large apples, which, after being well squeezed, afford from 35 to 40 per cent. of retorted gold. The retort commonly employed in quartz mills resembles in form and size a large black-lead crucible, furnished with a well-fitting cover, kept in its place either by a screw clamp, or by a clamp and wedge, and into which is screwed an inch iron pipe bent, with a gentle turn, at right angles, and, at a distance of about thirty inches, again so bent downwards as to form another angle.

Before using this retort, it must be slightly covered on the inside, by the use of a rag attached to a stick, with a paste made of water and clay, or sifted wood ashes, to prevent the adhesion of gold in case of too much heat being accidentally applied. The lumps of amalgam are now introduced, and the face of the lid carefully luted with a mixture of clay and wood ashes, after which it is placed on the retort, and securely fastened in its place by means of the clamp.

The crucible and its charge are now introduced into the furnace employed for melting retorted gold into bars, and which for this purpose may be fourteen inches square, and 1 ft. 8 in. deep above the fire bars. The end of the pipe will now be a short distance only from the floor of the furnace room, and beneath it is placed a vessel of water into which a piece of canvas, which is bound round it so as to form a short hose, is allowed to dip; the level of the liquid being constantly maintained by the flowing off of the water in the receiver, in proportion as the condensed mercury accumulates. The condensation of quicksilver is often effected by means of wet cloths bound around the descending limb of the pipe, and which are constantly kept cool by the application of water: the same result is sometimes more neatly and readily produced by the application of a Liebig's condenser.

When it has been thus arranged in the furnace, the fire may be lighted and gradually increased, until the retort has acquired a dull red heat, care being at the same time taken to effect the perfect condensation of the mercury by a constant supply of fresh water to the eduction pipe. The heat is thus kept up for several hours, according to the size of the retort and the amount of amalgam operated on; but the production of a decided light red heat, visible in daylight, should be avoided, since at that temperature the gold might not only be partially melted, but the retort itself seriously injured. When the pipe begins to get cool, and no more drops of quicksilver are observed to fall from its extremity, the operation is completed, and the fire may be withdrawn; but the cover should not be immediately removed, since a hot retort, even after the operation has been slowly and carefully conducted, gives off mercurial vapours, which would be, if inhaled, highly injurious. The water covering the mercury in the receiver, should, during a properly conducted retorting, remain perfectly clear and free from turbidity, since if it becomes milky it may be regarded as a proof that the heat employed has been too great. A forced retorting saves little or no time, and the mercury under such circumstances is imperfectly expelled, whilst the retort itself is rapidly destroyed, and soon rendered unserviceable. In order to obtain satisfactory results, the retort should be heated gradually, kept a long time at a black red heat, and allowed to cool before being opened. The fuel employed in the furnace is either coke or charcoal, and as soon as it has sufficiently cooled, the retort may be opened, and the spongy gold, which is of a light yellow colour, may be removed for the purpose of being melted into bars. When very large quantities of amalgam have to be dealt with, a fixed retort not unlike those employed in the manufacture of coal gas may be made use of, and, particularly when employed for the retorting of silver amalgam, is frequently made of a large size.

Melting Retorted Gold.—The fusion of retorted gold is commonly effected in the furnace employed for the distillation of amalgam in the small crucible-shaped retort. The melting is performed in black-lead crucibles, and the fuel employed, coke or charcoal. The crucible should be annealed by being gradually warmed, before being subjected to the full heat of the furnace, and a small quantity of borax is placed in it with the retorted gold. The gold from the retort being porous, occupies a considerable space, but after melting takes up much less room; so that as soon as the first charge has become fused, the cover may be removed and a further addition of retorted gold made. When the gold has become thoroughly fused, the crucible is withdrawn by the aid of a pair of strong tongs, furnished with jaws for enclosing and supporting its sides, and the metal is poured into moulds of cast iron. In order to render the handling of the pot perfectly safe, and to prevent any chance of the slipping of the tongs, its handles should be provided with a movable link, which, by keeping them tightly together, prevents the jaws from relaxing their hold on the crucible.

TABULAR STATEMENT.

OF THE OPERATIONS OF THE PRINCIPAL QUARTZ MILLS RUNNING IN CALIFORNIA IN THE YEAR 1861.—BY W, ASHBURNER,*

Name and Locality of Mill.	Water, or Steam.	No. of stamps.	Weight of each stamp.	No. of blows per minute.	No. of inches fall.	Horse-power developed by each stamp.	Height of screen above die.	No. of tons stamped per 24 hours.	Amount of wood consumed per ton.	Loss of mercury per ton.	Cost of extracting quartz from the Mine.	Yield per ton.	Cost of stamping per ton.	Total cost of treatment per ton.
			lbs.		in.		in.	tons.	cords.					
MARIPOSA COUNTY.														
Benton	water	64	550	60	[illegible]	1·00	5	80	...	0·0027	$5·53	$6·96	$0·559	$1·04
Bear Valley	steam	8	1000	65	12	1·97	...	20	0·156	0·007	6·00	25·24	1·618	2·10
Mount Ophir	steam	24	500	56	10	0·71	...	26	0·176	0·019	6·00	16·94	1·909	2·96
Princeton	steam	12	600	60	14	1·27	5	23	0·118	0·026	5·43	27·42	1·423	3·18
TUOLUMNE COUNTY.														
Union	water	20	500	60	13	0·98	...	20	...	0·031	13·00	50·00	3·145	3·31
Platt	steam	10	500	56	13	0·95	...	8½	0·130	0·080	5·00	25·36	2·820	2·63
Lomberdo's	water	8	650	58	14	1·32	...	15	...	0·016	8·00	20·00	1·243	1·64
Eureka	water	20	600	65	11	1·06	...	40	...	...	2·33	12·00	...	1·03
Confidence	steam	10	480	54	12	0·78	9	10	0·175	...	1·50	10·00	2·204	2·29
Telegraph	water	10	475	62	...	...	10	5	...	...	3·00	30·00	...	2·00
Knox and Co.	water	5	...	...	...	...	...	6	...	...	6·00	20·00	...	1·25
Yancy (Dry)	water	6	800	...	...	...	...	4	...	...	...	...	...	...
CALAVERAS COUNTY.														
Crystal (Dry)	steam	12	600	60	10	0·91	6	8	0·500	0·128	1·50	80·00	3·016	6·31
Angels Q. M. Co.	steam	16	490	50	...	0·62	10	24	0·130	0·125	2·00	5·00	1·827	2·03
Blue Wing (Dry)	steam	24	...	...	...	...	...	8	...	...	...	...	...	...
AMADOR COUNTY.														
Oneida	steam	16	...	52	10	...	...	16	0·250	0·089	...	10·50	2·351	2·64
Eureka	steam & water	40	599	80	10	1·19	6	60	0·066	0·064	2·50	10·25	0·963	1·32
Badger (Upper)	water	16	400	80	10	0·81	...	25	...	0·055	2·50	10·25	0·646	0·67
" (Lower)	water	12	400	80	10	0·81	...	20	...	0·057	2·50	10·25	0·759	0·78
Herbertville (Dry)	steam	30	500	65	12	0·96	...	14	0·071	0·114	4·50	20·00	3·043	4·50
American Q. M. Co.	steam	20	575	70	10	1·01	...	25	0·160	0·067	2·93	10·00	1·476	1·[illegible]
Spring Hill	steam	31	400 & 600	70	10	...	...	36	0·144	0·050	2·00	10·00	1·304	1·63
EL DORADO COUNTY.														
Empire (Logtown)	steam	10	600	60	13	1·18	6	12	0·145	0·054	2·75	10·00	0·900	1·49
Union (Aurum City)	steam	10	450	70	...	...	...	20	0·175	...	...	30·00	2·046	2·05
Tulles (Grizzly Flat)	water	11	1000·500	42	...	...	...	...	...	...	5·00	10·00	...	...
PLUMAS COUNTY.														
Eureka	water	28	...	...	...	...	...	...	...	...	3·73	15·45	...	6·27
Mammoth	water	12	500	40	8	0·65	...	10	...	0·028	...	25·00	...	...
SIERRA COUNTY.														
Sierra Buttes	water	24	600	50	10	0·76	9	30	...	0·027	5·67	14·82	0·633	1·36
Independence	water	12	550	48	9	0·60	8	10	...	0·033	5·00	15·00	0·558	2·90
NEVADA COUNTY.														
Nevada Q. M. Co.	water	12	850	60	12	1·54	3	15	...	0·198	4·00	17·14	1·466	2·65
Gold Hill	steam	21	730	52	14	1.84	1½	85	0·171	0·044	...	custom	1·900	2·30
Massachusetts Hill	steam	16	1000	55	12	1·67	4	33	0·183	0·024	26·00	70·00	...	2·91
Empire No. 1	steam		...	...	...	...			0·104	1·120	...	5·00	...	1·37
" No. 2	steam	6	900	62	14	1·97	5	18	0·141	0·109	10·00	26·00	...	1·66
Orleans	steam	9	600	32	10	0·48	2½	8	0·241	0·046	3·00		...	3·23
Allison Ranch	steam	8	950	55	13	1·71	5	17	0·176	0·023	20·00	50·00	1·678	3·60
Lady Franklin	steam	8	950	60	14	2·02	4	23	0·130	0·011	...		1·279	2·05
Forest Springs	water	10	800	60	...	...	4	8·76	...	0·044	15·00	52·35	1·343	2·96

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* Geological Survey of California, vol. i. p. 475.

The present average cost of treating gold quartz in California is estimated by Mr. Ashburner as follows:—

	Per ton of 2,000 lbs.
In Water Mills, water free	$1·22
„ „ purchased	1·60
In Steam Mills	2·14

CHAPTER X.

VEIN MINING IN HUNGARY AND BRAZIL—SODIUM AMALGAM.

AURIFEROUS VEINS AT SCHEMNITZ—STAMPING MILLS—HUNGARIAN BOWLS—SHAKING TABLES—CONCENTRATION OF AURIFEROUS SLIMES—GOLDLÜTTE SCHEIDETROG—RESULTS OBTAINED—MODE OF OCCURRENCE OF GOLD AT MORRO VELHO—STAMPING—STRAKES—TREATMENT OF FIRST TAILINGS—TREATMENT OF SECOND TAILINGS—AMALGAMATION OF CONCENTRATED ORES—LOSS OF GOLD AND MERCURY—COST OF WORKING—APPLICATION OF SODIUM AMALGAM—NOT GENERALLY EMPLOYED IN CALIFORNIA—ADVANTAGES STATED TO RESULT FROM ITS USE.

HUNGARY.—The metalliferous deposits of Schemnitz, in Lower Hungary, are generally regarded as being true veins, although they do not always exhibit distinct walls separating them from the enclosing rock, which is, however, in the neighbourhood of the veins, usually more or less decomposed, and often contains a considerable amount of iron pyrites. Some of these lodes are sixty feet in width, and have been worked to a depth exceeding two hundred and fifty fathoms. The veinstone is principally quartz, through which are disseminated, in a finely-divided state, galena, iron pyrites, gold, and sulphide of silver.

Stamping Mill.—These ores are treated in stamping mills, constructed according to the usual continental system.*

Each stem or lifter consists of a wooden beam, usually made of oak or beech, twelve feet in length and six inches square, the lower extremity of which is provided with a head of hard white cast iron. These are cast with a shank, which is inserted into the wood, and securely held in its place by two strong iron bands, driven on over the lifter and tightly wedged. The stamper thus prepared weighs two hundred and fifty pounds, although in some cases they do not exceed one hundred and twenty pounds in weight. The axle is of wood, nineteen inches in diameter, and provided with iron journals, the stampers being lifted by means of wooden cams and tappets. Each pestle has a lift varying from eight inches to a foot, and strikes, on an average, seventy blows per minute. The battery box is composed of thick planks, firmly put together; and the bottom on which the stampers fall is prepared by tightly beating in, with a heavy iron ram, fragments of hard quartz, which are introduced in successive layers.

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* Our data relative to the treatment in Hungary of gold ores by amalgamation, have been chiefly derived from a paper published in 1846, by M. Pache; and as we are without any precise information relative to the modifications which may have been since introduced, the descriptions of the various processes must be considered as applying to that date. The title of the paper referred to is as follows:—"Sur la Préparation mécanique des Minerais dans le district de Schemnitz (Basse Hongrie), par M. H. Pache."—*Ann. des Mines*, Tome x. série 4.

These mills are generally so constructed as to be self-feeding, although in some cases they receive their supply of ore from a small inclined hopper, into which the stuff to be stamped is thrown by manual labour. They have no gratings, and discharge the pulverised ore over the edge of an aperture running the whole length of the battery box, the height of which, from the face of the stampers, can be regulated by means of a sliding apron. The stamping is conducted wet, about three cubic feet of water per minute being supplied to each battery of five stampers.

The apparatus employed for the extraction of gold at Schemnitz is represented, Plate V. fig. 1, being an elevation, and fig. 2, a plan of the arrangement; B is the last battery of a range of stamping mills, A the axle, *b* the table before *flosh holes*, *c* the canal through which the stamped ore escapes from the battery box; *d*, *e*, *f*, are three canals or spouts at different levels, for the passage of slimes.

Hungarian Bowls, &c.—The eight amalgamating mills, 1, 2, 3, 4, and 1′, 2′, 3′, 4′, are disposed in two rows, of which the first is placed four inches above the other. The divisions *g* are seven inclined planes having the same flooring, but separated from each other by partitions two inches wide, and an inch and a half deep. The distance between these divisions is eighteen inches, and the length of the table six feet ten inches, with an inclination of fifteen inches between its two ends. Each of these inclined planes is completely covered over by two pieces of canvas a little wider than the platform, and slightly exceeding half its length.

The mills are composed of a cast iron basin, *a*, Plate V. fig. 3, half an inch thick in the metal, twenty-two inches in diameter at top, and seventeen and a half inches at bottom, with an internal depth of seven inches. This basin is simply placed, without any special foundation, on the floor, to which it is secured by means of two iron ears. The wooden cylinder *b* is bolted vertically in the centre of the pan, and is bound at top and bottom by iron rings, whilst at its upper extremity is an iron thimble, for the reception of the lower end of a vertical shaft. The runner, figs. 4, 5, 6, destined to revolve in this basin, is of wood, the pivot *c* of the spindle being so placed as to be exactly vertical, both with regard to the bottom of the basin and the face of the wooden block forming the revolving portion of the mill. This adjustment is readily managed by means of the tripod *e, f, g,* attached to the wood at its extremities, and of which the central boss *h* fits on to the square part of the spindle. The circumference of the wooden block *a′* is bound by two strong iron hoops, its lower face being provided with twelve blades or stirrers of thick sheet iron, driven into the wood, and projecting about half an inch, arranged as the radii of a circle. The space between the runner *a′* and the pan *a* receives 28 lbs. of mercury, the distance between the two surfaces being regulated by means of the nuts, *i, i′*. Formerly this was so arranged that the blades penetrated slightly beneath the surface of the quicksilver, but it has been ascertained that the amalgamation progresses more rapidly, and with less loss of mercury, when the lower edges of the iron stirrers are kept slightly above its level. The distance allowed between the circumference of the runner and the inside of the pan within which it revolves, is about one-sixth of an inch, and the upper extremity of each vertical shaft is retained by two bearings, K K′, fig. 1. These shafts have also a coupling in some portion of their length, which enables one or more of the runners to be thrown out of action, without interfering with the remainder.

Motion is communicated to the various mills by means of two mitre wheels, one of which is attached to the axle, whilst the other is keyed on a shaft on which is a pulley, *l*, around which is an endless strap bearing on the riggers of all the mill runners, as shown by the dotted lines. The relative diameters of the mitre wheels are so calculated as to cause the runners to make eighteen revolutions per minute. Higher and lower speeds have occasionally been given to these mills, but the above number of revolutions per minute has been found to afford the most satisfactory results. It has also been ascertained that from eight to nine mills are required for the treatment of the sands and slimes resulting from the action of every ten stampers, which amounts to about seven thousand pounds in the course of twenty-four hours. The working of this apparatus proceeds concurrently with that of the stamping mills, and requires but little supervision. The slimes are discharged into the canal or launder, *c*, from which they pass through a screen of wire gauze, *m*, by which any impurities, such as fragments of wood, are separated; and then flow into the canal *d*, from which they pass through four small spouts into the basin-shaped cavities in the runners of the mills 1, 2, 3, and 4. The slimes now descend so as to come in contact with the mercury contained in the basins, *a*, where they are constantly agitated by the iron blades fixed in the revolving wooden blocks, and finally rising through the annular cavity, between the edge of the runner and the inside of the basin, escape by the lip *n* into the mill immediately below it. Here precisely the same action takes place as in the first; and the slimes flowing off into the canal *e*, are distributed over the inclined tables, *g*, by means of apertures prepared for that purpose in the side of the launder.

Once every month the mills are thrown out of action, and the whole of the mercury removed. The quicksilver is now subjected to two successive filtrations, and an amalgam is obtained, which usually contains from twenty-eight to thirty-three per cent. of gold, and which is treated by distillation. The mercury separated by filtration, and which still retains a little gold, is returned to the pans, and the apparatus again put into operation. The monthly loss of mercury in an arrangement of eight mills, is usually from two and a half to three ounces.

One boy readily manages the supervision of from ten to a dozen of the tables, *g*, and every two hours removes the cloths from each in succession, and, after having washed them in a tank prepared for the purpose, replaces them as before. In the case of ordinary ores, affording from fourteen to fifteen loths per thousand quintals (about three dwt. per ton), eight loths are extracted by the mills, and from five to six by the inclined tables, so that from one to two loths only are obtained from the slimes by smelting. The sands and slimes passing off from the tables are collected in settling pits and labyrinths, from whence they are subsequently removed for concentration, principally on shaking tables.

Shaking Tables.—The shaking table employed at Schemnitz, which is somewhat peculiar in its construction, is represented Plate V. figs. 7, 8, 9, and 10; the first being an end, and the second a side elevation. The part *a* of the table is called its head, *b* is the tail, and *c* (figs. 9 and 10) is known as the tongue. Of the four chains by which the table is suspended, the two at the tail *b* are the longest, and are attached to the wooden rollers *d*, furnished at each end with ratchet wheels by means of which their length may be easily regulated. The wooden axle *e* is furnished with either two or three cams, which successively throw forward the lever *f*, articulated at *g*, carrying with it the rod *h*, which pushes the table in the same direction, as long as the cam continues to act on it, but, as soon as it is released, allows it to fall back with a sudden jerk, into its original position, the tongue *c* at the same time striking against a stout bar placed before two of the pillars of the framework for that purpose. The elasticity of this bar causes a series of vibrations in the table, which gradually become less, until it is again forced forward and thrown by the cam as before described. The position of the elastic bar forming the stop is easily regulated by wedges, whilst the length of the rod *h* admits of being adjusted by the use of the screw *i*, and wheel and pinion *k k'*, in such a way as to increase or diminish the throw of the apparatus. Above the axle of each table is a wooden platform, on which are stored the slimes to be treated, and which are from thence transferred to the cistern *l*, of which the bottom has a considerable inclination. Below the semicircular opening *m*, in this box, is a small spout or gutter *n*, leading to the higher end of the inclined plane *o*. This head-board is composed of two distinct planes, placed one above the other, but both inclined at the same angle, and on the upper one are arranged a number of little wooden buttons, *o'*, by the arrangement of which the regularity of the flow of slime over the whole surface is readily secured. In some cases, a strainer of wire gauze is placed across the head of the lower portion of the table head, for the purpose of collecting chips or any other impurities that may have accidentally become mixed with the slimes.

The two launders, *p* and *q*, run the whole length of the establishment, the first being used for the purpose of supplying clean water, and the second for carrying off the impoverished slimes.

These tables are set in motion by water-wheels, one horse power being sufficient to keep in operation ten tables, each of which is capable of washing the slimes produced by four stampers.

The working of these tables is conducted as follows:—Slime is thrown with a shovel into the cistern *l*, and a small stream of water allowed to fall into it, which, mingling with the slimes, escapes by the aperture *m*, and descending in an even flow over the inclined plane *o*, covers the whole width of the head of the table. In order to facilitate the mixture of the slimes and water, it is generally necessary to use a shovel, by which the contents of the box are occasionally stirred; and when the sands mixed with the slimes are somewhat coarse, a small wooden hoe is employed to rake the deposit, on the head of the table, gently against the current.

The tension of the table, or the sine of the angle formed by the suspension chains with the perpendicular, from the point at which they are attached, is made to vary in accordance with the nature of the slimes to be treated, although those of the lower end *b*, generally remain the same for all descriptions of slimes. The tension of these chains is generally about five and a half inches, but for the chains of the head *a*, it is not more than two inches for coarse slimes, whilst for very fine slimes it may be as much as eight inches. The extent of the throw of the table is also made to vary in accordance with the nature of the stuff to be wrought. For coarse slimes it may be as much as five and a half inches, whilst for very fine ones it is sometimes reduced to two inches. The number of oscillations of the table is so managed that it shall be constantly in motion, without any period of repose, the number of throws per minute, for coarse slimes, being about sixteen, and for very fine ones, from thirty-five to forty; but it must be remembered that in each case the elasticity of the bar against which the tongue is made to fall, produces from ten to fifteen secondary oscillations, gradually becoming less decided, until the table is again thrown forward by the cam.

After the apparatus has been in action a certain period, varying according to the nature of the slimes worked, from a few hours to two, or even three days, the deposit of enriched matter, towards the head of the table, accumulates to a depth of some six or seven inches, so that the operation could no longer be continued without danger of its flowing over the sides. The apparatus is then stopped for the purpose of being cleaned up; and as the deposit varies in richness in different parts of the table, it is divided into three distinct portions, each of which is differently disposed of. Even that at the head is, however, rarely sufficiently rich to be sent directly to the smelting works, and is generally again worked over. The deposit in the middle, which is divided into two classes, is invariably re-treated, whilst that at the lower end is usually rejected as useless. The different products thus obtained from the first table are further enriched by being subsequently re-treated on separate tables.

At Schemnitz, in addition to shaking tables, sleeping tables are sometimes employed for the concentration of the finer slimes, but these are of the usual construction, and are rather used for the purpose of washing lead slimes, than for those of which gold forms an important constituent.

Having described the amalgamating mills and shaking tables, it remains for us to follow the treatment of the rich auriferous materials collected on the cloths spread on the inclined tables, and subsequently accumulated in the tanks into which they are washed. The final washing of these rich deposits is a delicate operation, performed first in the gold box (goldlütte), and afterwards in a kind of scoop-shaped batea, called a scheidetrog.

Goldlütte.—This apparatus is represented, in plan and elevation, Plate V., figs. 11 and 12. It consists of an inclined trough, twelve feet in length, and twenty inches in width, with an inclination of about three feet on its entire length. Its end and vertical sides are nailed to the bottom, and are ten inches in depth. The upper portion of the plank forming the bottom of this arrangement has, cut in its surface, numerous small riffles, which form zigzags, running at an angle of 45° with the upright sides. At a distance of seven inches from the upper end of this box, is a dam *a*, rather shallower than the sides, behind which a stream of water is introduced by means of a tap, and which, flowing evenly over its upper edge, distributes itself regularly on the auriferous slimes placed at *b*. The bottom of this arrangement must be so fixed on the woodwork which supports it, that its transverse section is at all points perfectly level. The workman stands on the plank, *c*, and the three rectangular reservoirs, *d*, *e*, *f*, are for the purpose of receiving the products of the operation.

The working of this arrangement is conducted very much like that of an ordinary sleeping table. The ore is added in small quantities with a shovel, water is allowed to play over its surface so as to distribute it in the box, and then clean water alone is made to flow over it; whilst, at the same time, its surface is gently swept upwards, in a contrary direction to the current, with a brush or small heath broom. The products obtained are :—

1st. A coarse argentiferous slime containing gold, galena, and iron pyrites.
2nd. A fine slime containing the same substances.
3rd. A concentrated slime, rich in gold.

The two first products are usually re-washed in the same apparatus before being sent to the smelting works. The third, on the contrary, contains nearly the whole of the gold, and is at once washed in the scoop-shaped batea.

Scheidetrog.—This instrument is represented, in plan and elevation, Plate V., figs. 13 and 14. It is neatly cut out of a single block of wood, elm or sycamore being generally employed for the purpose. The workman holds this below the lower end of the washing box, and when in that position, the auriferous slimes are carefully brushed into it by the aid of a broom, assisted by a gentle current of water. He now holds it in both hands, placing his thumbs in the semicircular depressions *a a′*, and stationing himself before a tank of water, he begins giving to the instrument a motion similar to that of the shaking table, striking at each oscillation the end *b* against his thighs. After working in this way during a few minutes, the greater portion of the sulphides are washed off, and, in order to eliminate the remainder, he holds the batea in a slightly inclined position with the left hand, whilst with the right he directs on it a small stream of water from a horn in which a minute hole has been bored. This operation is commenced at the upper end of the instrument and gradually continued downward, by which means the gold, gradually freed from impurities, begins to make its appearance in the form of a pale yellow crescent-shaped streak. The product of each operation thus conducted is washed into a receiver, the contents of which are again re-washed, in the same way, at the close of the day's work. The final product thus obtained is amalgamated, by agitation with mercury, and after filtration in the usual way, retorted, and melted into bars.

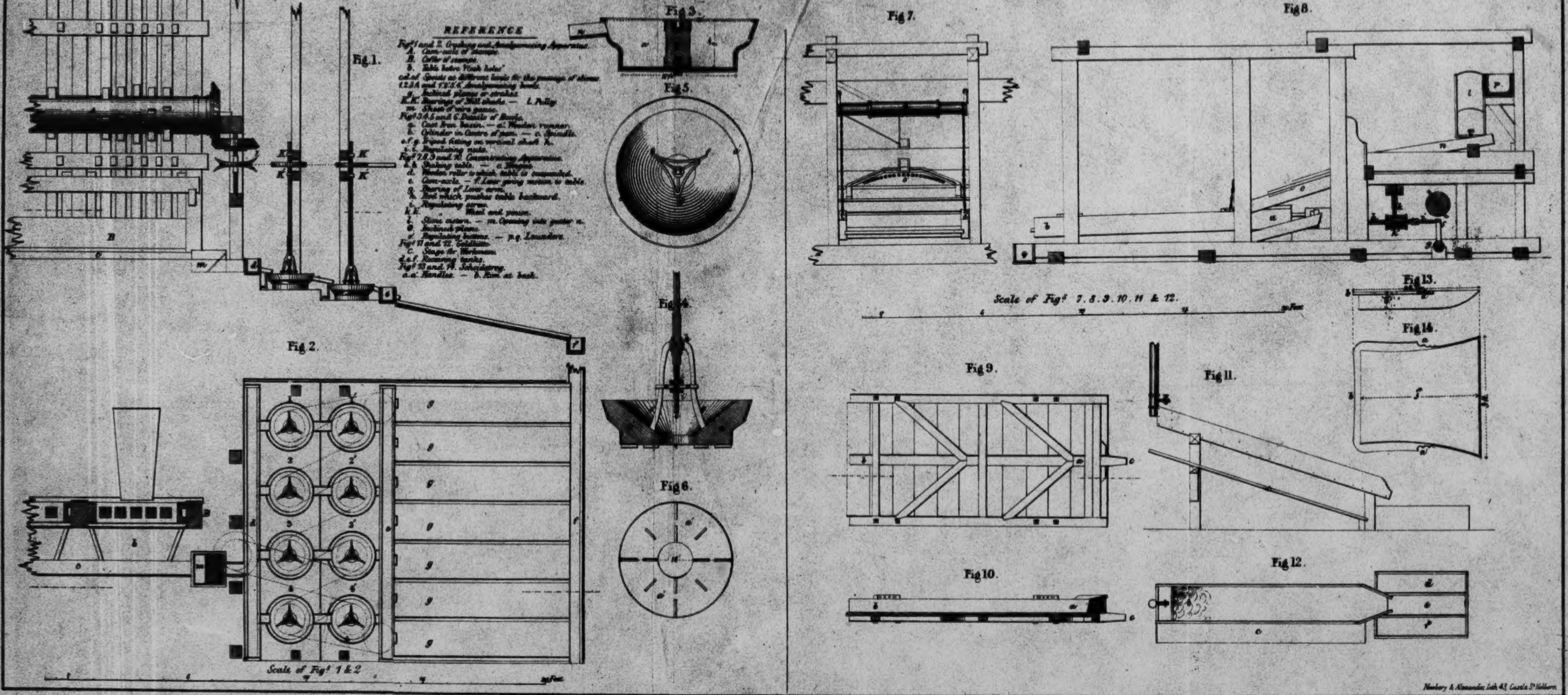
The relative values of the products obtained at Schemnitz from ores operated on in this way, after deducting the cost of metallurgical treatment, were, in 1846, nearly as follows:—

	Per cent.
Gold	50·5
Silver	33·0
Lead	16·5
Total	100·0

Brazil.—The rock treated at the Morro Velho mines is principally a mixture of magnetic, arsenical, and common iron pyrites, finely disseminated in, and intimately mixed with, a quartzose gangue. The composition of what is called pure ore may be taken at about forty-three per cent. of silica, and fifty-seven per cent. of pyritous matter. Of these minerals, arsenical pyrites is usually the most auriferous, although it does not occur in large quantities. Pure specimens of

PLATE V.

MACHINERY EMPLOYED FOR THE EXTRACTION OF GOLD AT SCHEMNITZ, LOWER HUNGARY.



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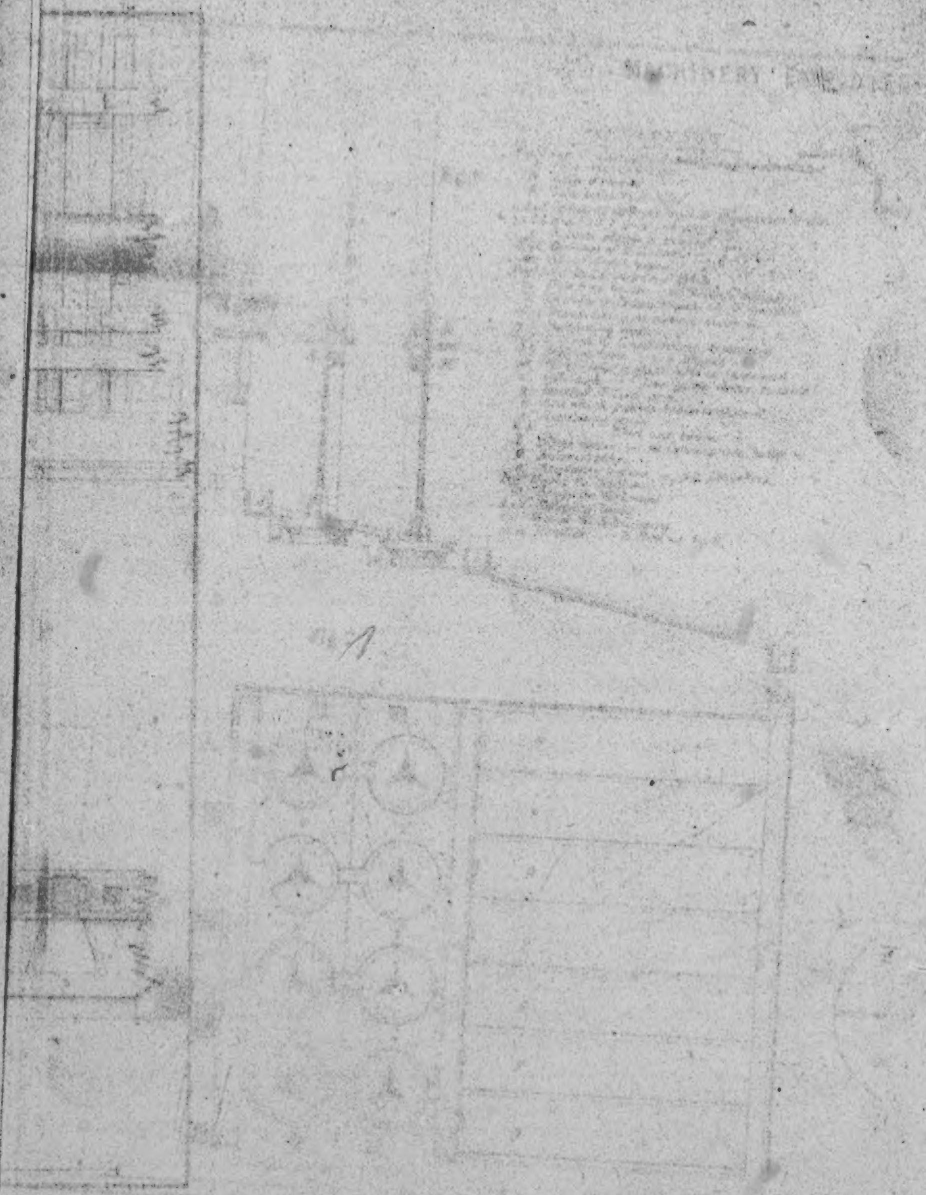
Figs 1 and 2. Crushing and Amalgamating Apparatus.
A. Cam-axle of Stamps.
B. Coffer of stamps.
b. Table before Wash hole.
c.d.e.f. Spouts at different levels for the passage of slimes.
1.2.3.4. and 1'.2'.3'.4'. Amalgamating bowls.
g. Inclined planes or strakes.
K.K. Bearings of Mill shafts — l. Pulley.
m. Sheet of wire gauze.
Figs 3.4.5 and 6 Details of Bowls.
a. Cast Iron basin. — a'. Wooden runner.
b. Cylinder in Centre of pan. — c. Spindle.
e.f.g. Tripod fitting on vertical shaft h.
i.i. Regulating nuts.
Figs 7.8.9 and 10. Concentrating Apparatus.
a.b. Shaking table. — c. Hinges.
d. Wooden roller to which table is suspended.
e. Cam-axle. — f. Lever giving motion to table.
g. Bearing of Lever arm.
h. Rod which pushes table backward.
i. Regulating screw.
k.k. Wheel and pinion.
l. Slime cistern. — m. Opening into gutter n.
o. Inclined plane.
d. Regulating buttons. — p.q. Launders.
Figs 11 and 12. Goldtische.
C. Stage for Workmen.
d.e.f. Receiving tanks.
Figs 13 and 14. Schaidtrog.
a.a'. Handles. — b. Rim at back.

Newbery & Alexander, Lith. 43 Castle St Holborn

E. & F.N. Spon. 48 Charing Cross, London.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this substance afford, by assay, from four to six ounces of gold per ton, and wherever crystals of this mineral make their appearance the yield of the precious metal is large. Cubical pyrites is of more frequent occurrence, but is far less rich in gold; solid specimens of this substance, but slightly mixed with quartz, yield about an ounce and a half of gold per ton by assay.

Magnetic pyrites constitutes the largest proportion of the sulphides found, but this is very slightly auriferous, since pure specimens generally yield rather less than four dwt. per ton. Branches of clay slate are often found in the principal veins, and this rock, under such circumstances, commonly affords, by assay, from five to seven and a half dwt. of gold per ton. Quartz without any admixture of sulphides has never been found to be auriferous, and it is a remarkable fact that the smallest speck of gold is rarely seen, previous to concentration, in any of the ores from this mine.*

Spalling.—The ores on being drawn from underground are delivered on the spalling floors, where they are broken into fragments of a proper size for the stamping mills; and the clay slate, and other comparatively sterile rock, are picked out and thrown into separate heaps. Iron hammers were, until 1857, employed for breaking the ore on the spalling floors, but in that year cast steel was substituted for iron, the result of which has been a saving of at least fifty per cent. on the annual cost of these tools.

Stamping.—The stamping mills employed at Morro Velho are worked by water, and are of the ordinary Cornish form of construction, except that each lifter is provided with four iron guides for keeping it in its proper position. A new stamp head weighs 230 lbs., and when entirely worn out is reduced to 59 lbs. The average weight of a stamp head may thus be taken at about 150 lbs., and the period of its duration at four months. The total weight of a new pestle, *i. e.* head, lifter, tongue, and all the other work attached, is about 640 lbs. The lift of the stamper is from ten to twelve inches, the number of blows struck per minute varying from 60 to 80, in accordance with the greater or less supply of water for the wheels.

The battery box, or coffer, differs slightly in dimensions in the different mills, but is composed of wood lined with sheet iron. The heads are generally worked about three inches from each other, and the same distance from the sides of the coffer. The height from the bottom of the coffer to the grate openings is a matter of great importance, and one that requires much attention, since on this greatly depends the degree of fineness to which the ore will be crushed, as well as the amount passed through the apparatus in the course of twenty-four hours.

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* We state this on the authority of Mr. Hockin, the Managing Director of the Company.

The stamping mills in this establishment are self-supplying, a blow being given to the feeding hopper, by a peg driven in the central lifter of each set, whenever the absence of ore in the coffer admits of its falling sufficiently low to come in contact with it. It therefore follows, as the stamps do not work on iron blocks, but form their own beds of tightly-compressed ore, that as the heads wear, the bottom of the coffer will grow higher, and the space between the grated openings and the faces of the heads become proportionately less.

In order to keep a series of stamping mills in a regular and efficient state of working, considerable attention is required to be bestowed on their adjustment; and when the bottom of the coffer has become too high, the working of the heads is so arranged, that before feeding themselves with fresh ore, they shall so far wear down the stamp bed as to reduce it to its proper level beneath the grate opening.

This adjustment is readily effected by taking out the pin from the middle lifter, and shifting it higher or lower, as the case may require.

It is probable that this operation may have given rise to the term applied to the regulation of stamps, which is called "pitching."

The grates employed at the St. John d'El Rey mines are nineteen inches in length, nine inches in width, and made of sheets of copper one-eighth of an inch in thickness. These are pierced with conical holes one-twelfth of an inch in diameter on the outside, tapering off on the other to one-forty-eighth of an inch, where a projecting burr is raised, which is placed toward the inside of the coffer. Copper plates have been found, on the whole, more durable than iron; besides which they possess the advantage, that when worn out they are readily converted into gun-metal by the addition of a little tin, and can then be employed for axle bearings and other purposes.

Results obtained.—The rock stamped at Morro Velho is reduced to the state of a mixture of fine sand and impalpable slime, the relative proportions of which will be seen from the following tabular statement of experiments:—

TABLE I.*

Size of Mesh.	Mixture from all the Mills.	From Herring's Stamps.	From Lyon's Stamps.
Of 1,000 grains sifted, on sieve of 10,000 holes per square inch—	per cent.	per cent.	per cent.
Passed through	88·25	95·25	93·22
Did not pass through	11·75	4·75	6·78
	100·00	100·00	100·00
Did not pass through 2,500 holes to square inch	0·50	0·25	0·33

This auriferous material issuing from the stamping mills is associated with gold in three different states, viz.—

1st. Free gold capable of concentration by washing.

2nd. Ditto in a lamellar form liable to be carried off in suspension by water.

3rd. Mechanically combined gold, enclosed in particles of pyrites, but capable of being liberated by further grinding.

The following table, from the notes of Mr. Dietzsch, affords much practical information relative to the stamping mills now in operation at this establishment:—

TABLE II.

Name of Stamps.	No. of Heads.	Blows per minute.	Lift of Stamp Heads.	No. of Cams on the axle.	Dimension of Stamps Coffer.						Dimension of Wheels.				* Ore stamped.		
					length.		width.		depth.		diameter.		width.		Tons per day.	Tons per head per day.	lbs. per head per day.
			in.		ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.			
Lyon. . .	30	63	10	...	2	2	1	3	2	0	40	6	3	3	36·73	1·22	2,733
Cotesworth.	12	61	11	4	2	2	1	1	1	6	31	0	3	0	16·92	1·41	3,158
Susannah .	9	65	12	6	2	3	1	3	2	0	19	2	4	4	10·94	1·21	2,710
Herring. .	24	78	12	...	2	6	1	6	2	0	42	6	6	0	34·56	1·44	3,225
Powles . .	36	67	12	4	2	2	1	3	1	10	51	0	5	0	66·10	1·83	4,100
Addison .	24	73	12	4	2	2	1	3	1	10	42	0	6	0	35·54	1·48	3,315
	135																

* From MS. notes of Mr. F. Dietzsch, Superintendent of Reduction Works.

Strakes.—The slimes and water issuing from the gratings in front of the coffers, to which a certain amount of clean water is added, are conducted over inclined platforms about eighteen inches in width, and from twenty-seven to thirty-five feet in length, which have a fall of one inch per foot. The first sixteen feet of these strakes are covered with bullock skins two feet two inches long, and of the width of the strake, tanned with the hair on them; a series of baize cloths, each two feet ten inches long, are employed below these, which are again followed, at the lower end of the arrangement, by another series of overlapping skins. A certain quantity of finely-divided gold is caught even on the last of these skins, whilst a considerable proportion, estimated at ten per cent. of the total amount present, is carried off in suspension by the water.

The skins and strips of baize are removed and washed at regular intervals during the day and night, the time allowed to intervene between each washing up, depending on the nature and richness of the ores treated. In each of the houses erected over the strakes, are boxes or tanks, in which the skins and pieces of baize are carefully beaten and washed, and of which those destined for the reception of the most concentrated slimes are divided by partitions into three separate divisions. Into one of them the first three skins from the upper end of the strakes are washed, and into the second is removed the ore collected on skins 4 and 5 of the series, whilst the third division contains water in which the final washing of skins 1, 2, 3, 4, and 5, is effected, after having previously removed the coarser particles in one of the other divisions.

The skins and cloths below No. 5 are washed in the same way, but into separate tanks, from which they are subsequently removed and washed over another series of skins and cloths, in order to effect their further concentration. The deposit on the first three skins, known as *head sand*, amounts to about 0·42 of a cubic foot per ton of ore stamped, and contains from twenty-seven to thirty ounces of gold per ton, all of which, with the exception of about one ounce, is in a free state. This sand is sent without further preparation to the amalgamating house.

The *middle sand* consists of the deposit collected on the skins Nos. 4 and 5, and contains about six ounces of gold per ton, of which some sixteen dwt. only are mechanically combined with particles of pyrites. This sand is further enriched by being washed over another system of strakes.

The deposit taken from the third compartment of the tank in which the upper five skins are finally washed, is called *swim sand*, and being exceedingly fine, could not be safely subjected to concentration, and is consequently sent, with the head sand, directly to the amalgamating house.

All the products caught below the fifth skin are known as *tail sands*, and are like the middle sand, concentrated by being washed over a second system of strakes. The apparatus employed for this purpose consists of three strakes, each covered with four skins and seven cloths.

At the Herring, Lyon, Powles, and Addison stamps, the skins and cloths are changed every hour, whilst at the others this is only done every two hours. Previous to 1854 the strakes above described constituted the only arrangement for the collection of the auriferous materials issuing from the batteries, but in that year some extra strakes were added, as it had been long known that a considerable amount of the precious metal passed off in the slimes. These extra strakes are so fixed as to receive the slimes from the lower end of the original ones, after being first diluted by the addition of some clean water, and the sands collected are subjected to amalgamation without further concentration. The saving of gold effected by these additional strakes amounts to about 38 ounces per month.

The valuable nature of the products resulting from the above processes of concentration, renders it necessary that all parts of the establishment in which they are collected should be protected from theft, and for this purpose the strake heads are securely railed off, and the tanks in which the head sands are washed provided with locked covers.

In Californian mills, where the amount of blanket surface employed is proportionately much less than at Morro Velho, that part of the establishment which contains the various contrivances for the separation of gold from the crushed rock, is securely enclosed in a wooden building, under the immediate control of the chief amalgamator. No part of the gold-saving apparatus is, as a rule, kept under lock, with the exception of the tanks, when full, in which the blankets are washed, and the rollers with the mercury boxes of the amalgamators, which are frequently secured by locked covers.

The following table gives some important details relating to the strakes employed at the Morro Velho mines:—

TABLE III.

Name of Stamps.	No. of Heads.	No. of Strakes.	Length of Strakes.		Width of Strakes.		Area of Strakes, square feet.	Square Feet per Ton of Ore stamped.	Tons of Ore passing per 24 hours.	No. of Skins on Strakes.	No. of Baizes on Strakes.	Quantity of Head Sand per day.
			ft.	in.	ft.	in.						cub. ft.
Lyon . .	30	36	31	10	1	6	1,719	46·83	36·73	288	210	20·00
Cotesworth	12	13	30	6	1	4½	545	32·21	16·92	104	65	5·50
Susannah .	9	8	27	0	1	6	324	29·61	10·94	48	48	2·75
Herring .	24	29	35	0	1	6	1,232	35·64	34·56	228	174	18·00
Powles. .	36	42	33	7	1	3½	1,821	27·55	66·10	336	252	28·47
Addison .	24	30	31	10	1	5	1,352	38·04	35·54	240	170	17·00
	135	158					6,993		200·79	1,244	919	91·72

Mr. Dietzsch remarks that straking may, on the whole, be considered a cheap, simple, and economical process, by which 67 per cent. of the gold originally present in the ore is obtained in a highly concentrated state, whilst the 33 per cent. which escapes is in two distinct forms.

1st. Light free gold.

2nd. Gold enclosed in the coarser particles of pyrites.

The first, which has, to a great extent, been laminated by the action of the stampers, exposes so great an amount of surface, in proportion to its weight, that it cannot be economically saved by any known process, and floats off with the lighter portions of the slimes. It is difficult to determine the exact amount of loss resulting from this cause, but from experiments which have been made on the residual sands, it is believed to be about 10 per cent. of the gold present in the original ore. The second, which is in the form of fine gold enclosed within particles of pyrites, is partially recovered by separating the coarser from the finer sands, and subsequently grinding them in arrastres, and subjecting the resulting secondary slimes to the action of another system of strakes.

Re-treatment of First Tailings.—This is effected by running the whole of the waste from the different systems of strakes above described, through ordinary tyes, similar in construction to those represented Plate IV. c c′, in which small wooden stops are from time to time placed at the end, thus allowing the slimes to run off, whilst the coarser sands remain behind. These tyes are arranged in couples, so that whilst one is being cleaned out, the other may be in operation.

When one of these tyes has been thus filled, the sand from it is taken out into another box, from which it is conveyed by a stream of water into arrastres, which have an opening in the side about a foot from the bottom. Into these arrastres, of which there is a series, driven by water power, the coarse sands are constantly flowing, mixed with a stream of water; and becoming ground under the mullers, are so reduced in size as to float, and then escaping through the apertures in the sides, are concentrated on a system of strakes. It is evident that the action of these machines will be continuous, since the supply of sand to be pulverised is constantly flowing in, and being heavier than the finer particles, it will sink to the bottom, where it remains until it has been ground into an impalpable state, when it escapes in suspension over the strakes.

Each arrastre is supplied with three cubic feet of water per minute, each cubic foot holding in suspension 1·75 lbs. of sand, and discharges over three strakes, 15 feet in length, covered with either five or six skins. The quantity of sand passed through in the course of twenty-four hours, is about three tons. The deposits collected on the first skin are regarded as head sands, and washed off every two hours into tanks, from which they are sent directly to the amalgamating house. These sands usually contain about 16½ dwt. of gold per ton. The deposit on the second skin is removed every four hours, and, like that on the first, subjected to direct amalgamation, whilst the other skins are only washed every twelve hours, and the slimes afterwards subjected to concentration.

Below the first series of strakes, a number of extra strakes are laid down, which are worked precisely like those attached to the first system before the stamping mills.

Re-treatment of Second Tailings.—Up to the year 1855, the tailings from the system of apparatus before described were treated as worthless, and sluiced into the river accordingly, although it had been determined, by numerous experiments, that they still retained a notable amount of gold. At that period, however, a distinct system of works was erected on the banks of the river, and at a low level, for the elaboration of the whole of the refuse escaping from the upper floors.

These are provided with a series of tyes, in which the sands are separated from the slimes, and from which they are subsequently taken to be again ground in stamping mills and arrastres.* The sand from these tyes contains about 1Γ dwt. of gold per ton.

As these sands could not be advantageously ground in stamping mills, without the admixture of some coarse and hard material, they were for some years stamped with the addition of a portion of a deposit, known as "Cascalho," containing small quantities of gold; but in 1861, the supply of this substance having become to a great extent exhausted, the refuse portions of the lode, picked out at the spalling floors, were substituted for it. The result of this experiment proved highly satisfactory, and consequently the employment of these wastes, which contain a small amount of gold, has been since continued.

The system of enrichment on strakes is conducted precisely as in the case of the upper floors, and about 16 cubic feet of concentrated sand is daily produced. These sands, however, contain large quantities of decomposing sulphides, giving rise to the formation of sulphates of iron, which rapidly flour and sicken any mercury brought in contact with them, and the tailings are consequently not adapted for direct amalgamation. They are therefore again further concentrated, by being passed over another set of strakes, until they have become reduced from 16 to 1½ cubic feet, and are then carefully washed in a batea. This batea is a shallow wooden bowl, used in the same way as the tin dish of the Californian miner, and in which some six or eight pounds of sand are operated on at a time. The product of this last washing chiefly consists of finely-divided gold, associated with small quantities of various mineral impurities, from which it is separated by trituration with mercury in large Wedgewood-ware mortars.

Amalgamation of Concentrated Ores.—The apparatus employed at Morro Velho is represented Plate VI. In this drawing fig. 1 shows the machinery in elevation, and fig. 2 in plan, whilst fig. 3 is a vertical section of the triturator, or "saxe."

The water-wheel A, by means of intermediate gearing, gives motion to a set of barrels H, 4 feet in length, and 2 feet 5 inches in diameter, each having a capacity of 20 cubic feet. The charge of each barrel is 16 cubic feet, or one and a half ton of wet sand, and 60 lbs. of mercury. There is also a sufficient amount of clean water at the same time introduced, to give to the slimes the necessary degree of fluidity to enable the globules of quicksilver formed to become properly incorporated, without allowing them to become sufficiently mobile to admit of the settling of the mercury and amalgam at the bottom.

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* The stamping mills employed at this part of the establishment have no grates, but, like those of Schemnitz, discharge once over sluices or dams.

The barrels, when thus charged, are allowed to revolve in accordance with the nature of the ores treated, and the state of the atmosphere at the time, during from thirty to thirty-six hours; and the bungs L, being then removed in succession, the contents of the barrels are discharged into the hoppers M. The stop-gates *a* are now slightly raised, and a small stream of water is introduced into the hoppers, which gradually washes their contents into the spouts N, where an additional supply of clean water is added from the launder F, and the united streams, carrying with them in suspension a portion of the slimes from M, fall into the transverse conduit O, from whence they pass through the funnels *b*, into the trough, or saxe W.

The saxe consists of a trough, sixteen feet in length, and two feet in width, divided into six compartments by the partitions *w*, and provided with a movable agitator, or runner V, which can be removed at pleasure by being vertically drawn out of the trough W. This is supplied with rakes, fixed on the faces of solid wooden blocks *w′*, corresponding to the number of divisions in the saxe, and supported on a bar, running on guide wheels, in such a way, that the depth of the rakes can be easily regulated by means of screws; a certain amount of play being at the same time allowed them by the introduction of springs.

Motion is given to the rakes by the eccentric Q, and the rods S, U, and bell-crank T, the rod U being at one end connected with the movable bar of the runner.

The working of this apparatus is conducted as follows:—The contents of the barrels are first discharged into the hoppers M, and from thence slowly washed, by a constant stream of water, into the saxe through the funnels *b*, traversing the two central runner-blocks, and discharging beneath the surface of the mercury, whilst the rakes are slowly drawn backwards and forwards, by the action of the eccentric Q, over the surface of the mercury, which accumulates in the trough of the saxe W.

In this way the globules of quicksilver and amalgam are deposited from the diluted slimes, in the bottom of the various divisions, while the slimes themselves flow regularly through the apertures in the divisions, shown by the dotted lines, fig. 3, and, escaping at each end, pass over strakes Y, covered in the usual way with bullock skins, by which a large portion of the mercury, which might otherwise be lost, is collected. It will be observed, by referring to fig. 3, that the central division of the saxe is not, like the others, provided with an aperture; and consequently the slimes entering by one funnel are discharged to the right, while those arriving by the other escape to the left of the apparatus. Extra strakes are arranged below the first, which are worked with the addition of clean water, as in the system of supplementary strakes employed below the stamping mills, and the supply of clean water admitted on the strakes Y is regulated by means of the stop-cocks X. The usual loss of gold per ton of ore treated is, at the St. John d'El Rey mines, estimated at about 2 oitavas or nearly 5 dwt. per ton. The mercury can be drawn off when required, and the amalgam is cleaned up, strained, and retorted every ten days, the gold being then melted into bars.

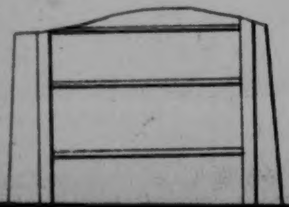
The loss of quicksilver, taken on the average of the last three years, has been 2·923 oz. per ton of ore stamped. The average cost of extracting the mineral from the mine, and its reduction, including every expense incurred by the Company for general management, &c. &c., for the last ten years, has been 25*s.* per ton; out of which the cost of stamping alone has been 2*s.* 10*d.* per ton.

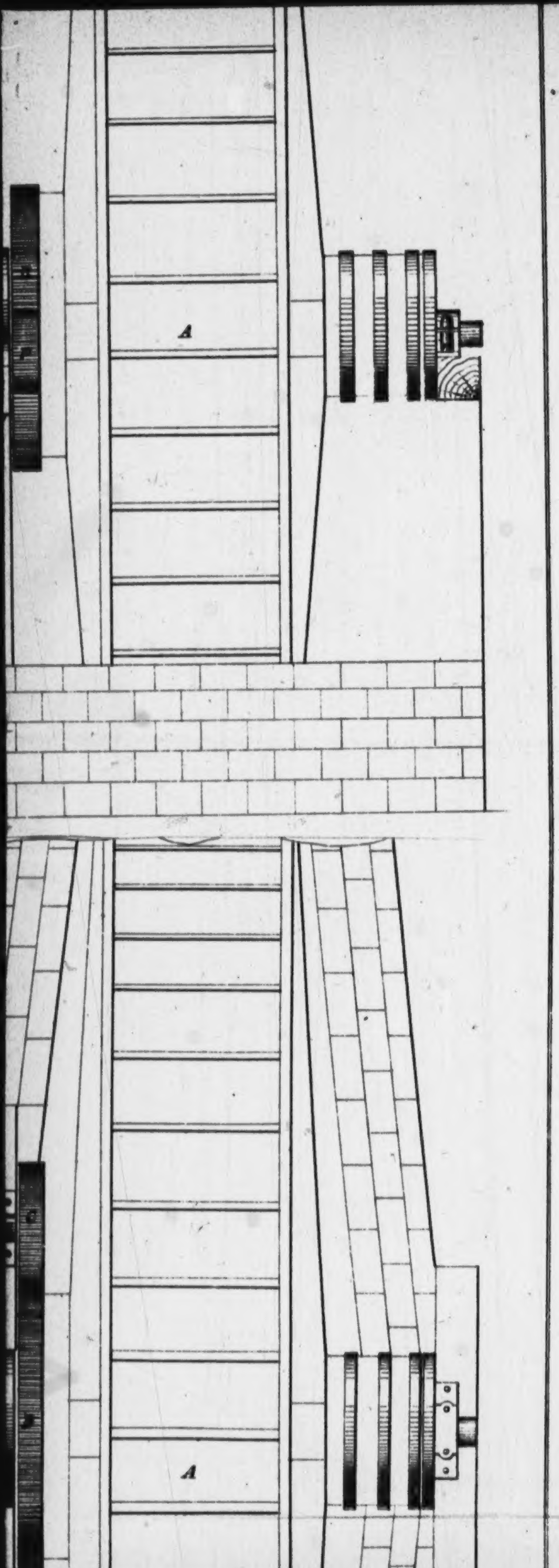
SODIUM AMALGAM.

The extraction of gold by amalgamation has, hitherto, been often attended with difficulties occasioned by the presence of compounds of sulphur, arsenic, antimony, bismuth, or tellurium in the ores; which by covering the gold with a thin film of tarnish, prevents its entering into combination with mercury. The use of sodium amalgam, however, is stated not only to facilitate amalgamation under such circumstances, but also to prevent the "sickening" of mercury, which in the presence of certain chemical compounds, and among others sulphate of iron, is apt to take place. It is also claimed for sodium amalgam that by its use the "flouring" of mercury when ground with compounds containing sulphur, arsenic, tellurium, &c. may to a great extent be avoided.

Two claimants dispute the honour of having first discovered these properties of an amalgam of sodium and mercury, but the truth would appear to be that the discovery was made by each, almost at the same time, and without any knowledge of what was being done by the

PLATE VI.

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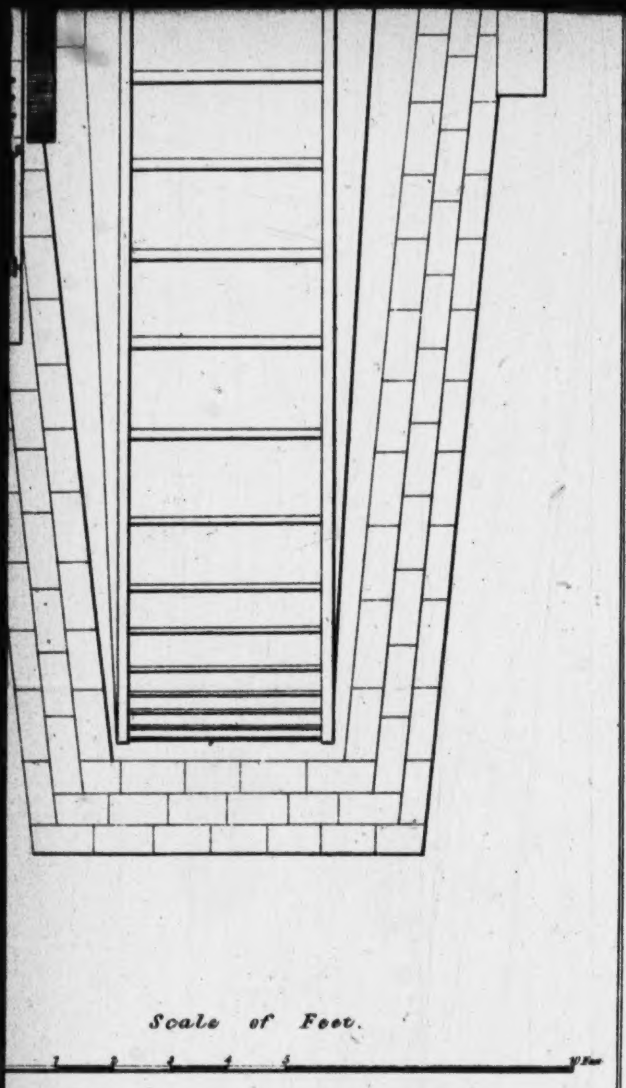
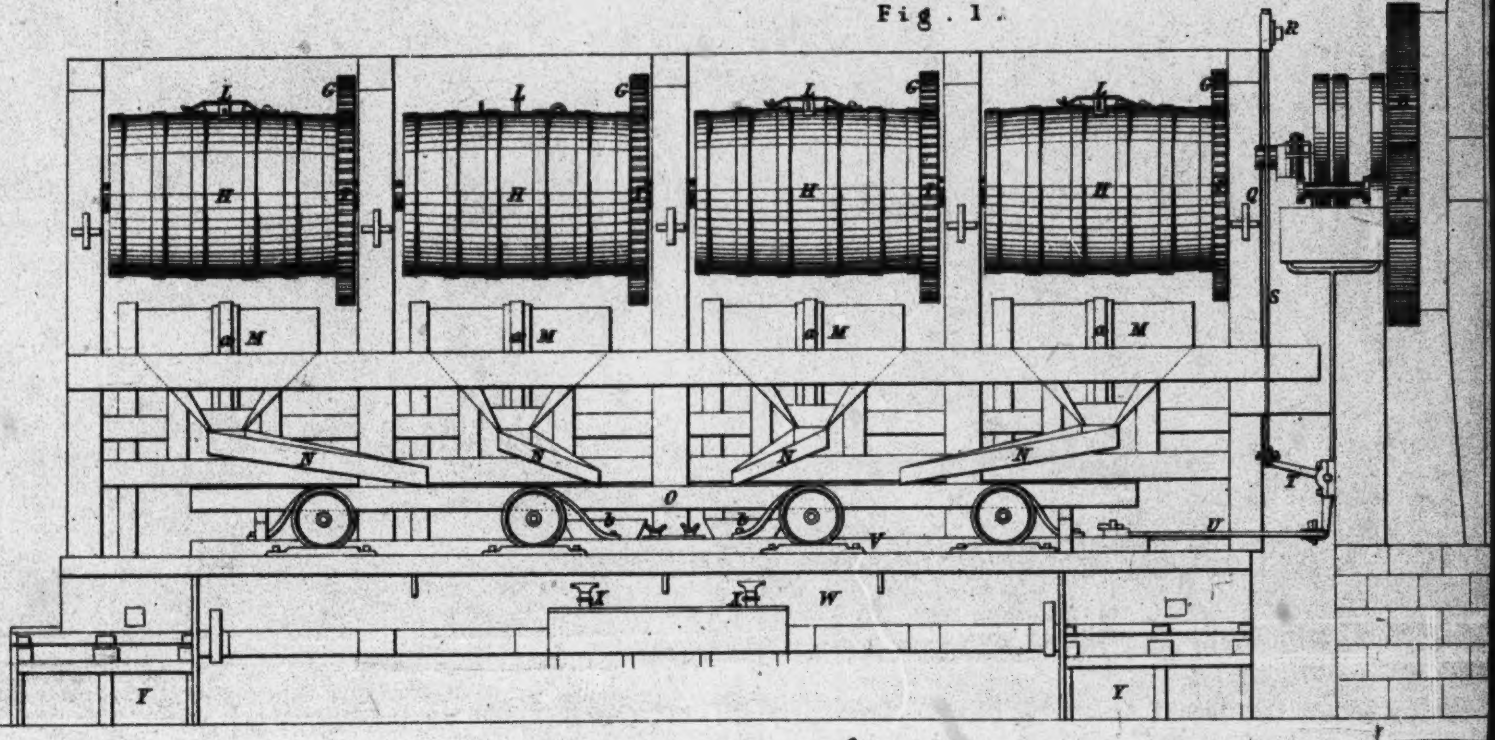


Newbery & Alexander, Lithl. 43 Castle St. Holborn.

AMALGAMATING MACHINERY
AT
MORRO VELHO MINE.

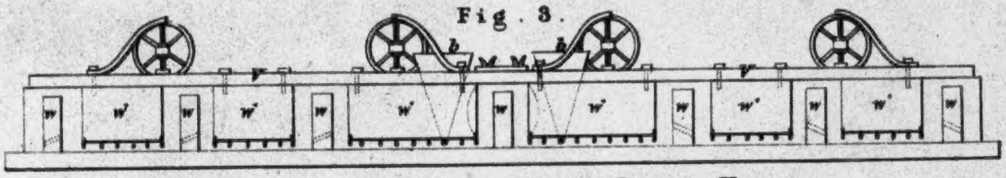
AT
MORRO VELHO MINE.

Fig. 1.



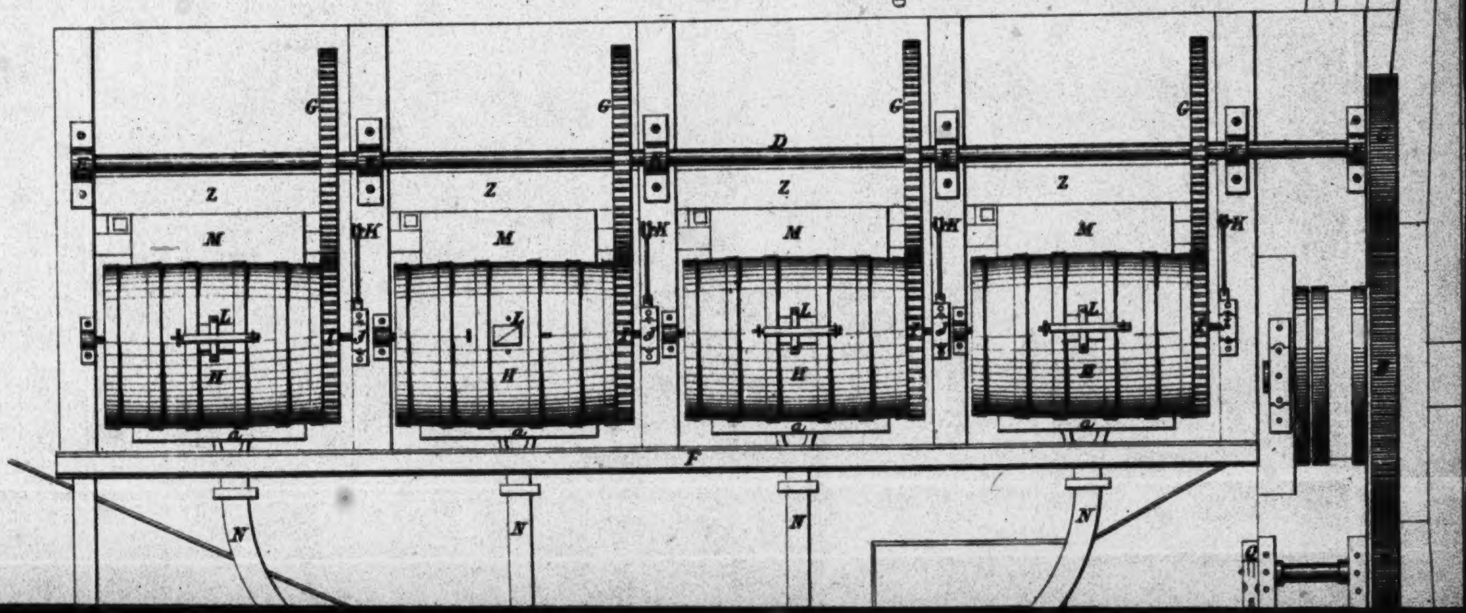
Section

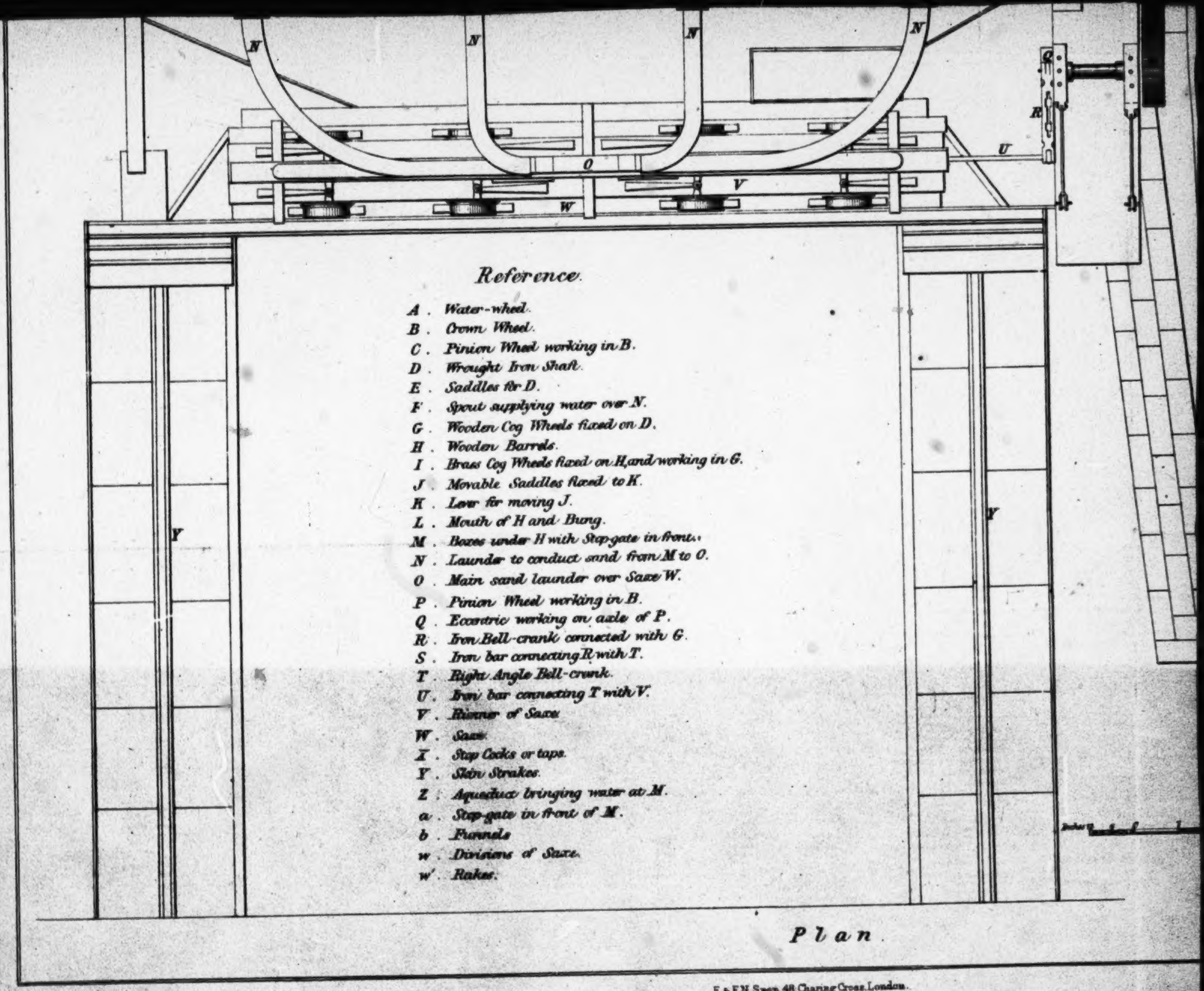
Fig. 3.



Section of the Saxe W and Runner V.

Fig. 2.





Reference.

A . Water-wheel.
B . Crown Wheel.
C . Pinion Wheel working in B.
D . Wrought Iron Shaft.
E . Saddles for D.
F . Spout supplying water over N.
G . Wooden Cog Wheels fixed on D.
H . Wooden Barrels.
I . Brass Cog Wheels fixed on H, and working in G.
J . Movable Saddles fixed to K.
K . Lever for moving J.
L . Mouth of H and Bung.
M . Boxes under H with Stop-gate in front.
N . Launder to conduct sand from M to O.
O . Main sand launder over Saxe W.
P . Pinion Wheel working in B.
Q . Eccentric working on axle of P.
R . Iron Bell-crank connected with G.
S . Iron bar connecting R with T.
T . Right Angle Bell-crank.
U . Iron bar connecting T with V.
V . Runner of Saxe.
W . Saxe.
X . Stop Cocks or taps.
Y . Skin Strakes.
Z . Aqueduct bringing water at M.
a . Stop-gate in front of M.
b . Funnels
w . Divisions of Saxe.
w' . Rakes.

Plan.

E. & F. N. Spon. 48. Charing Cross. London.

other. Dr. Wurtz of New York applied for an American patent in November 1864, whilst Mr. Crookes made a similar application in England in February 1865, and consequently before anything was known in Europe of the existence of the American patent. It must, however, be conceded that the first application for a patent was undoubtedly made by Dr. Wurtz.

Not having any personal experience of the effects produced by the addition of sodium amalgam to mercury employed for the amalgamation of gold ores on a large scale, the following description of its action is given on the authority of the inventors of the process, and has been chiefly derived from an article by Dr. Wurtz, published in the *American Journal of Science and Arts*, vol. xli. March 1866, and from a pamphlet on the same subject, published for private circulation, by Mr. Crookes. It may be remarked that at the date of our leaving the mining districts of California in December 1866, although its use had been extensively experimented on, it had not become generally adopted, and that the evidence with regard to its efficiency was considered to be of a conflicting nature. It may, nevertheless, be regarded as certain, that no time was lost in practically testing the efficiency of sodium amalgam, and that, had it possessed all the advantages claimed, its use would have long since become general.

A quantity of sodium amalgam dissolved in a hundred times or more its weight of quicksilver, is said to communicate to the whole a greatly enhanced power of adhering to metals, and particularly to those which, like gold and silver, are situated towards the negative extremity of the electro-chemical scale. This power of adhesion in the case of the two metals is so great, that the resistance which their surfaces, when in their native state, often oppose to amalgamation (a resistance much greater, and more general than has been hitherto recognised, and due to causes as yet uninvestigated) is instantly overcome, whether their particles be coarse or impalpable. Even an artificial coating of oil or grease, which is usually such an enemy to the combination of mercury with other metals, forms no obstacle to immediate amalgamation by this prepared quicksilver. The atoms of quicksilver are, as it is described, put into a sort of polaric condition, by a minute addition of one of the metals which range themselves toward the electro-positive end of the scale; so that its affinity for the more electro-negative metals is stated to be so greatly exalted that it seizes upon, and is instantaneously absorbed by their surfaces, just as water is absorbed by a lump of sugar, or other porous substance soluble in it.

Such quicksilver even adheres strongly to surfaces of iron, steel, platinum, aluminium, and antimony; an adhesion which, however, in the case of these metals is not a true amalgamation, there being no penetration into the substance of the metal; so that the superficially adherent quicksilver may be readily wiped off, just as water may be removed from glass: the only metal as yet experimented on, which cannot be enfilmed by the use of sodium amalgam, appears to be magnesium.

Application of Sodium Amalgam to working Ores of the Precious Metals.—This consists in adding from time to time, to the quicksilver used in amalgamation, about one hundredth part of its weight of sodium amalgam. The frequency with which the amalgam is to be added cannot be exactly specified, as it will be found to depend on a multitude of circumstances, such, for instance, as the temperature, the purity and quantity of the water used, the ratio borne by the surface of the quicksilver to its mass, the amount and mode of agitation of the quicksilver, the nature of the process and apparatus used, the character of the ore, strength of the amalgam, &c.; so that this important point can only be determined in each case by experience. Some general indications may, however, be derived from the experiments which have been made. It is said that less sodium is requisite in cases in which much water is employed, and when the water is frequently renewed, as, for instance, in the riffles of a sluice, and in all forms of amalgamators through which a continual current of water is kept running; since mercurial solutions of sodium are but little affected by water free from acid, alkaline, or saline impurities. In cases, however, in which but little water is employed, and especially where the ore and quicksilver are ground together into a slime, the water soon becomes alkaline, and oxidation of the sodium sets in, necessitating its frequent renewal. In such cases the following manipulation is recommended. The whole amount of quicksilver to be used for working up a batch of slimes, say fifty pounds, is prepared by dissolving in it one per cent. of amalgam No. 2; or better, two per cent. of the soft amalgam No. 1, which dissolves more readily:[*] one-half, or twenty-five pounds, is then thrown into the mill with the ore, and as the incorporation proceeds; certain fractional parts of the other half are added at intervals, varying according to circumstances, until the whole has been introduced. If, as is usual, the quicksilver has been separated from the slimes of a previous operation, it will retain a certain amount of sodium, and therefore require fresh amalgam in proportionately smaller quantities.

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[*] No. 1 amalgam contains two per cent., and No. 2, four per cent. of sodium, and is a hard brittle solid, remarkably infusible, requiring a temperature nearly as high as the fusing point of type metal to melt it, and may be cast into ingots, and packed either under petroleum, or in air-tight iron cans filled with dry lime.

In sluicing operations, the soft amalgam No. 1 is, on account of its ready solubility in mercury, most recommended; and in these cases it is practicable to test the quicksilver in the riffles, and ascertain when the magnetic quality requires restoration, by throwing in a few grains of gold dust.* Similar tests are easily applied to slimes, and in amalgamating generally, a slip of tarnished sheet copper is a suitable agent for such testings.

It may be remarked that the amalgam No. 1 is at any time easily prepared from No. 2, by melting it in an iron ladle with its own weight of quicksilver. In copper-plate amalgamation,—that is, in cases in which auriferous materials are brought into contact with amalgamated metallic surfaces,—it is recommended to substitute for quicksilver itself the pasty amalgam No. 2. In these modes of amalgamation great economy in wear and tear of apparatus, as well as in first cost, is said to be effected by using plates or surfaces of iron instead of copper. The power of coating or enfilming iron is stated to render these amalgams peculiarly valuable in every form of apparatus for amalgamation, which has internal surfaces of iron; for these becoming coated with quicksilver immensely extend its chances of contact with particles of gold, so fine as to remain suspended in the water.

Other important services are expected by the inventors to arise out of this power of enfilming iron, such as keeping the surfaces of stamps and of other apparatus used in crushing ores continually coated. In like manner, as the power of adhesion of quicksilver to other metals is exalted by the presence of the alkali-metals, so also is its own cohesion stated to be greatly increased. It is rendered more difficult to mechanically divide, and when thus divided again runs instantly together upon contact. Hence new results of great value are said to have been obtained. For instance, the so-called flouring or granulation of quicksilver, which in the amalgamation of ores always occasions losses both of the quicksilver itself, and of its amalgams with the precious metals, is stated to be reduced to a minimum, or altogether prevented.

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* From the attraction which it appears to possess for other metals, Dr. Wurtz calls the compound of sodium and quicksilver, "Magnetic Amalgam."

The recovery of floured quicksilver and amalgams from slimes and similar mixtures, is also said to be greatly facilitated and accelerated thereby. For this purpose some sodium amalgam is thrown into the separator, and collects and incorporates all the scattered globules of auriferous amalgam.

It is here necessary to call attention to a method of manipulation generally applicable when sodium amalgams are used, and particularly so in all cases in which the ore is ground or agitated with quicksilver in contact with metallic iron. This arises from the liability of abraded particles of iron to adhere to the amalgam.

The following plan is therefore in such cases recommended. The amalgam, after separation from excess of quicksilver, and before retorting, is fused in an earthen dish or iron ladle, with, if necessary, addition of a little quicksilver to make it more liquid; and the iron, which forms a scum on the surface, is skimmed off. The excess of quicksilver may, after cooling, be again separated from the amalgam in the usual way. Any amalgam which adheres to the iron scum is readily detached by boiling in water to remove the sodium. This process depends on the fact that adhesion to the iron totally disappears with the extraction of the last traces of sodium from the quicksilver. It is in fact possible to remove all iron from the amalgam by boiling in water without any previous fusion, particularly if the water be made somewhat acid or alkaline. The presence of iron can be readily detected by the magnet, which may also be sometimes used with advantage in separating iron from amalgam after all the sodium has been extracted. There are still other substances which may be found adherent to the amalgam when sodium has been used, such as platinum and osmiridium. These, like iron, immediately detach themselves on the removal of the sodium by boiling the amalgam in water. A mixture of platinum, or osmiridium, or both, with iron, may be freed from the latter by the magnet.

The sodium amalgams prepared in this country in accordance with the recipes of Mr. Crookes, and sold by his agents, are known respectively as A, B, and C amalgams.

Each of these contains three per cent. of sodium, in addition to which B has a small quantity of zinc in its composition, and C a little tin. An amalgam (A), of seven times the strength of the above, is prepared in solid bars for shipment when the expense of freight or land carriage is great. Amalgams B and C cannot be prepared in the concentrated form. It is recommended that one part by weight of amalgam B or C be dissolved in thirty parts of the mercury which is to be used in the amalgamating, triturating, or grinding machines, and the effect which it produces on the mercury noted from time to time during the operation. If it retain its fluidity and brightness to the end of the operation, it is a sign either that a sufficient amount, or too much has been added, and a second experiment should be tried with a less quantity of amalgam. But if it be floured, or sickened, or any loss occur, more amalgam may be added until the best proportion is arrived at.

Mr. Crookes states that amalgam B will generally be found effective, but if the ore contain an excess of any mineral which has a deleterious action on mercury, more especially if it contain bismuth, it will be advantageous to employ amalgam C instead of B.

When the best proportion of amalgam B or C is determined, small quantities of amalgam A should be introduced into the mercury, already containing amalgam B or C, in the proportion of one part of amalgam A to one thousand of mercury. This quantity of amalgam A can be added every few hours, according to circumstances, but one charge of amalgam B or C will, it is stated, usually be sufficient for several days. Under some circumstances it will be found advisable to add amalgam B or C every few days, but a little experience, and comparison with the results obtained by the old plan, will soon show how these several agents are best utilised.

CHAPTER XI.

ASSAY OF AURIFEROUS ORES—ESTIMATION OF GOLD CONTAINED IN QUARTZ—REFINING—ASSAY OF GOLD BULLION.

ASSAY OF GOLD QUARTZ—FUSION WITH LITHARGE OR RED LEAD—AURIFEROUS PYRITES — CUPELLATION — INQUARTATION — PARTING — ASSAY TABLE — TABLE SHOWING PROPORTION OF GOLD IN AURIFEROUS QUARTZ—REFINING—PLATINUM VESSELS—CAST IRON PANS—ASSAY OF BULLION—QUANTITY OF LEAD NECESSARY FOR CUPELLATION OF ALLOYS OF GOLD AND COPPER—ASSAY LABORATORY—THE TOUCHSTONE.

Assay of Auriferous Ores.—Although the exact amount of gold contained in a given specimen of rock is readily ascertained, it is often much more difficult to obtain a fair average sample of the whole produce of a vein. When the gold is in a finely-divided state, and equally disseminated throughout the rock, this presents comparatively little difficulty; but when, on the contrary, it is granular, and occurs in pockets and irregular deposits, great care and labour are sometimes necessary in order to obtain representative samples of the mass.

It is, therefore, of the highest importance that whenever rock is to be assayed for gold, the greatest care should be observed in preparing the samples on which the operation is to be subsequently conducted. With this view, the rock, of which it is desired to ascertain the yield, should be broken, and the heap cut through in the usual way, at least a ton being taken from each pile of importance, and reduced to fragments of the size of beans. This may be done, where machinery for dry crushing is not at hand, by means of a flat-faced hammer on an iron plate. After well mixing, this pounded ore is again cut through, and about ten pounds of it taken for the purpose of being still further reduced in size, and passed through a sieve of fine wire gauze; on this last from three to six different assays are made, and their mean result taken as the produce of the rock examined. By operating in this way a great degree of accuracy may be ensured; but where a less amount of exactitude is required, the quantities of rock crushed may be somewhat reduced, and the number of assays made, fewer.

Instead of operating as above described, a given weight of finely-powdered rock, from the last cutting, may be reduced in volume by careful washing in a batea, and the residue dried, and assayed in the usual way. From the results obtained, the amount of gold present in a ton of original rock can be readily calculated.

In establishments for the crushing and amalgamation of auriferous rock, it is, however, generally of more importance to determine the amount of gold retained by the tailings, than to ascertain by assay the actual produce of the original ore; since the quantity yielded by amalgamation on the large scale, added to that present in the resulting sands and slimes, will, for many practical purposes, sufficiently represent the total contents of the rock. It is also desirable to know exactly the amount of gold carried off in the final tailings, in order, not only to be enabled to judge of the effect of any modifications which may have been introduced in the machinery or methods of working, but also for the purpose of ascertaining if due attention has been exercised during the various stages of the different processes to which the ore has been subjected. When required for this purpose, the tailings should be caught at regular intervals throughout the day, by placing a bucket, or some other convenient vessel, under the spout from which they finally make their escape; and when full of water and sand, it must be placed aside, in order to give the matters in suspension time to settle. As soon as the water in the bucket has deposited the whole of the solid matter which it held in suspension, it is carefully decanted off, and the deposit, accumulated at the bottom, removed into a box prepared for that purpose.

This is repeated at intervals of about two hours, and at the termination of the experiment the contents of the box are dried, thoroughly mixed and submitted to assay. In the case of samples of tailings thus obtained, the discrepancies between the several assays will, in general, not be very considerable, since the coarse particles of gold will, as a rule, have been retained by the blankets and other appliances over which the stamped rock has passed, and the remaining fine gold is disseminated with a considerable degree of regularity throughout the mass.

Fusion with Litharge or Red Lead.—When the rock to be operated on does not contain an appreciable quantity of iron pyrites, or any other metallic sulphide, weigh 600 gr. in a finely-pulverised state, and intimately mix it with 4,000 gr. of litharge or red lead, and from 15 to 20 gr. of flour, starch, or finely-powdered charcoal.* Introduce this into a crucible, of which it should occupy about one-half the capacity, and heat it in an ordinary assay furnace, until the whole has entered into a state of perfect fusion, when the crucible is withdrawn and allowed to cool. When sufficiently cold, it is broken, and the button of lead extracted, cleaned, and cupelled. In the case of ores containing a large amount of gold, it is far better to break the crucible than to pour its contents into an iron mould; but in works in which tailings are being daily examined, the practice of pouring assays may be adopted with advantage, as from the smallness of the quantity of gold contained in the material operated on, no appreciable error will be the result, whilst, on the other hand, the saving in crucibles will be considerable.

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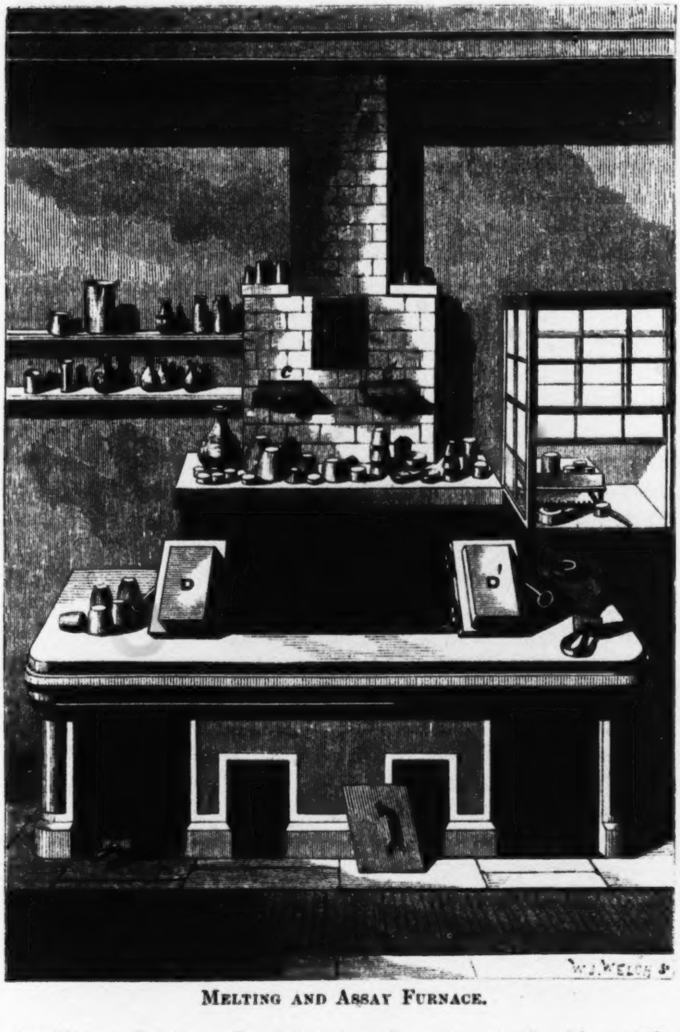
* The weight of pounded ore is fixed at 600 grains, because this quantity can be easily fused in an ordinary No. 10 French crucible.

In quartz-crushing establishments generally, the assays of ores and tailings may be conveniently conducted in the furnace employed for retorting and melting, which is generally of sufficient size to admit of three or four fusions being carried on at the same time. In regular metallurgical laboratories, for the sake of durability, and to prevent the cracking of the brickwork, the outside of the melting furnace is usually secured by iron plates as shown Fig. 37, which represents that employed by Mr. F. Claudet, and in which A A′ are the fireplaces, B B′ the fire-bars, and *b b′* the ash-pits. The dampers C C′ permit of regulating the draught, and the mouths of the furnaces are closed by the hinged doors D D′, lined with baffle-plates. Instead of hinged doors, sliding plates are sometimes employed, and are, for general purposes, probably preferable. The dimensions of this furnace are, 10 inches square and 16 inches in depth above the fire bars, which can, when necessary, be drawn out from the front for the purpose of allowing the coke to fall into the ash-pits *b b′*, or for unclinkering the grate. When used with charcoal, the crucibles must be supported on the bars on pieces of firebrick; but when coke is employed, it has in itself sufficient resistance to allow of their being embedded in the fuel without any other support.

The best crucibles for this purpose are ordinary French pots, which, before being introduced into the fireplace, should be annealed by being gradually heated on the top of the furnace, in order to expel any moisture they may have imbibed, and which, if they were too suddenly heated, might cause them to crack.

When, in addition to gold, the rock contains iron pyrites or any other sulphurised mineral, it frequently happens that the admixture of charcoal becomes unnecessary, and the fusion may be made with litharge alone, since one part of iron pyrites reduces eight and a half parts, blende seven parts, and sulphide of antimony, or grey copper ore, about six parts of lead, to the metallic state.

FIG. 37.



MELTING AND ASSAY FURNACE.

Auriferous Pyrites.—To determine the amount of gold contained in quartz much mixed with iron pyrites or other metallic sulphides, or to assay auriferous pyrites resulting from the concentration of tailings, the following process is generally resorted to. The amount of the substance on which it has been determined to operate, is weighed into a shallow scorifier, and then introduced into the mouth of the muffle of the cupelling furnace, care being taken not to heat it at first too strongly. The powdered ore is from time to time stirred with a bent iron wire fitted into a wooden handle, and in proportion as the sulphur is evolved, and the substance becomes less fusible, the scorifier is pushed further into the muffle, and consequently subjected to a higher temperature; the stirring, by means of the iron rod, being at the same time continued. The roasting must thus be carried on until all odour of sulphur has ceased to be evolved, and the scorifier with its contents is then withdrawn, and allowed to cool.

When sufficiently cold, the roasted ore is united with six times its weight of litharge, its own weight of dry borax, and from fifteen to twenty grains of charcoal, and the whole fused in the furnace as above described. In making the addition of the reducing agent it is necessary to add such a quantity as will afford a button of lead sufficiently large for the collection of the whole of the gold present, and at the same time of a size convenient for cupellation, since, if the amount of lead were too small, a loss of gold would be the result, whilst a very heavy button would require a large cupel, and occupy a long time in working off.*

Cupellation.—This process is founded on the circumstance that when an alloy of lead and silver, lead and gold, or of lead, silver, and gold, is exposed in a state of fusion to the action of a current of air, the precious metals are neither perceptibly volatilised, nor become oxidised, whilst the lead rapidly absorbs oxygen with the formation of a readily-fusible oxide. In order, therefore, to obtain the gold and silver contained in the assay buttons, it is merely necessary to expose them, on a porous fire-proof support, to such a temperature as will cause the oxidation of the lead, whilst the precious metals are not so affected. The litharge thus produced is absorbed by the porous body on which the assay is supported, and nothing but a button of gold or silver, or of an alloy of the two, in case of their being both present, ultimately remains on the test. These supports, which are called *cupels*, are made of bone-ash slightly moistened with water, and consolidated, by pressure in an iron mould, into the required form.

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* Instead of operating as above, the assay may be made by scorification.—See Assay of Silver Ores.

Cupels should be perfectly dry before being used, and ought to be kept for at least a fortnight in a warm place in the laboratory, previous to being employed for making cupellations. The muffle, which is made of baked fireclay, has the form of a small D-shaped retort, closed at one end, and is generally provided with openings in the sides and ends, in order to allow of the circulation of air through it.

The furnaces used for cupellation differ considerably in shape and method of construction, but the muffle, when fixed, is so arranged that whilst its closed extremity is supported by a proper shelf, the other corresponds with an opening in the front of the furnace, to the sides of which it is luted with a little clay, and has before it a small platform, on which the hot cupels can be allowed to stand when withdrawn from the muffle.* The muffle thus placed can be equally heated in every part, whilst the apertures, in the sides and end, admit of the passage through it of a current of air from the mouth, through the interior of the muffle, into the furnace itself. To light this apparatus, a little ignited charcoal or coke is thrown into the furnace, which is afterwards filled with fuel; and as soon as the muffle has become heated to bright redness, six or eight cupels, that have been annealing at the mouth of the opening, are introduced by means of proper tongs; the bottom of the muffle having been previously covered with a little loose bone-ash to prevent its being attacked in case of any litharge passing through the cupels in the course of subsequent operations. The open end of the muffle is now closed by a tile or door provided for that purpose, to prevent the introduction of cold air, and the cupels are raised to the temperature of the muffle itself. When this takes place, the door is removed, and a button, obtained from the fusion with litharge, introduced into each of the cupels by a pair of slender tongs. The mouth of the muffle may now be again closed for about a minute, in order to facilitate the fusion of the alloy; and on its removal, each of the cupels will be found to contain a bright metallic bath, in which state the assay is said to be *uncovered*. Under these circumstances, the lead is rapidly converted into litharge, which is absorbed by the cupel as quickly as it is formed, whilst at the same time a little white vapour rises from the cupels, and is gradually carried off through the end, and through the openings in the sides. A circular stain is at the same time formed around the metal in the cupels, which gradually extends and penetrates into their substance. When nearly the whole of the lead has been thus removed, the remaining bead of rich alloy appears to become agitated by a rapid circular motion, which seems to make it revolve with great rapidity. At this stage, the rotation of the bead will be observed to cease, and the button, having for an instant emitted a bright flash of light, becomes suddenly immovable. This is called the *brightening*, and a button remains on the cupel, composed of an alloy of gold, with the silver derived from the litharge, added to that which may have originally existed in the ore operated on.

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* The best fuel for the cupelling furnace is hard coke broken into small pieces.

When the proportion of silver is, in comparison with that of gold, very considerable, and the button of alloy is itself a large one, the abrupt removal of the cupel from the muffle might cause the metallic bead to spirt or *vegetate*, by which a portion of the metal might be thrown off, and a certain amount of loss thereby entailed. To prevent this taking place when a large button has been obtained, the cupel must be gradually withdrawn towards the mouth of the muffle; or the cupel in which the assay has brightened, may be covered by another, kept hot for the purpose, and then removed to the small platform in front of the muffle.

From the fact that silver becomes sensibly volatile at very elevated temperatures, it becomes necessary, when this metal is present, to make the cupellations at the lowest possible heat at which they can be effected. The temperature best fitted for this operation is obtained when the muffle is at a full red heat and the vapours which arise from the assays curl gradually away, and are finally removed by the draught. When the muffle is heated to whiteness, and vapours rise to the top of the arch, the heat is too great; and when, on the contrary, the fumes lie over the bottom, and the sides of the openings in the muffle begin to darken, a little more fuel must be added, or the draught increased. Should the temperature fall below this point, and the lead *set* in the cupels, they may generally be again started by closing the mouth of the muffle with a few large pieces of charcoal. These should be removed when the complete fusion of the alloy in the cupels has been again effected. If an assay has been properly conducted, the button obtained is round, bright, and smooth on its upper surface, and is readily removed from the cupel.

Inquartation.—In order to be enabled to completely remove silver from an alloy of that metal with gold, by the use of nitric acid, it is essential that the weight of the silver present should be about three times that of the gold. It is consequently necessary that when the amount of gold in the button of lead is approximately known, a piece of pure silver should be added to it before cupellation, of such a weight, that with the quantity of that metal derived from the litharge (ascertained by a separate assay) it will, as nearly as possible, fulfil these conditions. The only inconvenience, however, attending the addition of too large a proportion of silver is, that the gold obtained by the subsequent action of acid is thereby rendered flocculent, and somewhat more difficult to collect. When the object of the assay is to ascertain only the amount of gold present, without estimating the associated silver, the necessary addition of this metal is best made by placing a piece of the required weight on the leaden button obtained from the fusion with litharge, and driving it into the lead by a blow with a bright-faced hammer. In the case of poor ores, containing less than 10 dwt. of gold per ton, the silver derived from the litharge will of itself frequently be sufficient for the purpose of inquartation, whilst, as before stated, for the estimation of gold in richer ores, the addition of a little pure silver before placing the leaden button in the cupel becomes necessary.

When the ore treated contains silver in addition to gold, and it is desirable to ascertain its amount, it becomes necessary to first cupel the button of lead without the addition of silver: the metallic globule thus obtained is weighed, and its weight noted, as is also that of the amount of silver derived from the reduced litharge. If necessary for the operation of parting, more silver is now added by folding the globule obtained from the cupel, together with a bit of silver, in lead foil, and again passing it to the cupel. Lastly, the button of alloy resulting from the second cupellation, is dissolved in nitric acid, and the gold weighed. The amount of silver present in the ore will consequently be represented by the weight of the button of alloy obtained from the first cupellation, less the united weights of the gold, and of the silver yielded by the reduced litharge.

Parting.—The button obtained by cupellation is first allowed to cool, squeezed laterally between the jaws of a pair of strong pliers, and brushed with a hard brush to remove any adhering litharge, and then flattened by a bright-faced hammer on a steel anvil. After being carefully examined, in order to ascertain that it is perfectly free from extraneous matter, the flattened button is taken between the points of a pair of steel forceps, and dropped either into a long-necked flask of about two ounces' capacity, or into a large test-tube, and about half an ounce of nitric acid of 22° Baumé = 1·18 sp. gr. perfectly free from any trace of hydrochloric acid, added. The acid must now be boiled over a spirit lamp or gas jet, until all action on the alloy has ceased, and the liquid carefully decanted off. About a quarter of an ounce of pure nitric acid of 32° B. = 1·28 sp. gr. is then introduced, and boiled for about ten minutes, when it is, in its turn, poured off, and the flask or tube filled with distilled water. Its mouth is now covered by a small crucible, and its position reversed in such a way, that the gold which it contains may fall through the water into the crucible beneath it; and when this has taken place, its mouth is raised to the surface of the water in the crucible, and by a sidelong movement withdrawn altogether from it. The water is now carefully decanted off, and the crucible laid in a hot place to dry, after which it is heated to redness in the muffle, and when cold the gold is removed and weighed. Instead of using a flask, a thin crucible of porcelain may be employed for the operation.

Weighing.—The balance employed for assays of gold and silver should weigh to one-thousandth of a grain, and be provided with a set of decimal grain weights. A good balance should be fitted with an apparatus for steadying the pans, connected with the axis which moves the beam, in such a way that one movement of the handle first releases the beam, and subsequently the pans. The knife-edges should be made of agate, so that all the working parts may be enabled to resist injury from the fumes of the laboratory, or the effects of a damp climate.

The beam of such a balance ought not to be less than eight inches in length, and must be divided for the use of a sliding weight moved by an apparatus provided for the purpose. The whole is enclosed in a glass case supported by adjusting screws for placing the instrument level. To prevent wearing, the weights should be handled by the aid of a pair of forceps provided with ivory points.

From the result obtained by weighing the gold yielded by assay, the amount contained in a ton of ore of 20 cwt. may be readily calculated, either directly or by the aid of the following table :—

TABLE

*Showing, from the weight of Metal obtained from an Assay on 600 grains, the amount of Gold in oz. dwt. and gr. contained in a ton of Ore.**

If 600 grains of Ore give Fine Metal,	One ton of Ore will yield Fine Metal,			If 600 grains of Ore give Fine Metal,	One ton of Ore will yield Fine Metal.		
gr.	oz.	dwt.	gr.	gr.	oz.	dwt.	gr.
0·001	0	1	2	0·100	5	8	21
0·002	0	2	4	0·200	10	17	18
0·003	0	3	6	0·300	16	6	16
0·004	0	4	8	0·400	21	15	13
0·005	0	5	10	0·500	27	4	10
0·006	0	6	12	0·600	32	13	8
0·007	0	7	14	0·700	38	2	5
0·008	0	8	17	0·800	43	11	2
0·009	0	9	19	0·900	49	0	0
0·010	0	10	18	1·000	54	8	21
0·020	1	1	18	2·000	108	17	18
0·030	1	12	16	3·000	163	6	16
0·040	2	3	13	4·000	217	15	13
0·050	2	14	10	5·000	272	4	11
0·060	3	5	8	6·000	326	13	8
0·070	3	16	5	7·000	381	2	5
0·080	4	7	2	8·000	435	11	2
0·090	4	18	0	9·000	490	0	0

TO FIND THE WEIGHT OF GOLD IN A MIXTURE OF THAT METAL AND QUARTZ.—In purchasing fragments of rich gold quartz, it is sometimes desirable to be enabled to ascertain their intrinsic value, without separating the gold, and thereby destroying the specimen. From the

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* To use this table, write beneath each other, on a slip of paper, the number of oz. dwt. and gr. corresponding to the several decimal figures obtained by weighing, and add them together for the yield per ton of ore. As an example, let it be supposed that the gold resulting from an assay, on 600 gr. of ore, weighs 0·235 gr. On consulting the table we find—

gr.		oz.	dwt.	gr.
0·200	. . . .	10	17	18
0·030	. . . .	1	12	16
0·005	. . . .	0	5	10
		12	15	20

as the yield of gold per ton of ore.

known specific gravities of gold and quartz, it becomes easy, after determining the density of the mass, to ascertain approximately the amount of each present in any particular specimen.

The following Table exhibits the proportion, by weight, of gold in auriferous quartz of a given specific gravity; the formula from which it was calculated being found as follows:—

Let g be the specific gravity of gold (taken as 19).
q „ „ of quartz (taken as 2·6).
m „ „ of the unknown mixture.
x the unknown proportionate *bulk* of gold, to a unit of bulk of the mixture.

Then xg is the proportionate weight of gold, and $(1-x)q$ is the proportionate weight of quartz, and of course the sum of these must give the weight of the unit of mixture. That is to say, $xg+(1-x)q=m$. This is a simple equation, from which we find $x=\frac{m-q}{g-q}$, for the proportionate *bulk* of the gold. The proportionate *weight* is $\frac{gx}{m}$, or $\frac{g}{g-q}\times\frac{m-q}{m}$.

The formula may be verified as follows. Let y be the proportionate weight of the gold. Then $\frac{y}{g}$ is its bulk, and $\frac{1-y}{q}$ the bulk of the quartz. Hence the bulk of the mixture $\frac{1}{m}=\frac{y}{g}+\frac{1-y}{q}$, which gives $y=\frac{g(m-q)}{m(g-q)}$ as before.

With the values given above, $\frac{g}{g-q}=\frac{19{\cdot}0}{16{\cdot}4}=1{\cdot}1585366$; and the logarithm of the fraction is 0·06391.

The table is read thus: Auriferous quartz, whose specific gravity is 5·2, contains 0·5793 of its weight of gold; or, which is the same thing, 10,000 ounces of the quartz contain 5·793 ounces of gold.

TABLE

Showing the proportionate weight of Gold in a mass of auriferous Quartz, when the specific gravity of the mass is known.

Specific Gravity.	Proportion of Gold.	Specific Gravity.	Proportion of Gold.	Specific Gravity.	Proportion of Gold.	Specific Gravity.	Proportion of Gold.
2·6	0·0000	3·7	0·3444	5·2	0·5793	8·5	0·8042
2·65	0·0219	3·8	0·3659	5·4	0·6007	9·0	0·8239
2·7	0·0429	3·9	0·3862	5·6	0·6206	9·5	0·8415
2·75	0·0632	4·0	0·4055	5·8	0·6392	10·0	0·8573
2·8	0·0828	4·1	0·4239	6·0	0·6565	10·5	0·8717
2·85	0·1016	4·2	0·4413	6·2	0·6727	11·0	0·8847
2·9	0·1198	4·3	0·4580	6·4	0·6879	11·5	0·8966
2·95	0·1375	4·4	0·4739	6·6	0·7021	12·0	0·9075
3·0	0·1545	4·5	0·4892	6·8	0·7156	13·0	0·9268
3·1	0·1869	4·6	0·5037	7·0	0·7282	14·0	0·9434
3·2	0·2172	4·7	0·5176	7·2	0·7402	15·0	0·9577
3·3	0·2458	4·8	0·5310	7·4	0·7515	16·0	0·9703
3·4	0·2726	4·9	0·5438	7·6	0·7622	17·0	0·9813
3·5	0·2979	5·0	0·5561	7·8	0·7724	18·0	0·9912
3·6	0·3218	5·1	0·5679	8·0	0·7820	19·0	1·0000

REFINING.—SEPARATION OF GOLD AND SILVER.—The silver produced from some of the mines of the American continent, ancient European money, old Mexican and Peruvian dollars, and bars obtained from goldsmiths' sweep, are composed of silver which usually contains a certain proportion of both copper and gold. The processes for the separation of these metals from each other have been rendered so inexpensive and simple, that bars containing but half a thousandth of gold can now be refined at a profit.*

The auriferous silver is first melted, and granulated by being poured into water, and then attacked by boiling in a platinum vessel with from two to two and a half times its weight of sulphuric acid of 66° Baumé. Each attack is made on about 80 lbs. of the granulated alloy, and occupies from three to four hours; the operation being considered finished when effervescence has ceased, and sulphuric acid begins to be evolved.

The attack of the granulated alloy by sulphuric acid is necessarily attended with a copious evolution of sulphurous acid gas, which is generally allowed to escape into the atmosphere, but may be conducted into a sulphuric acid chamber, and thus utilised.

Before passing into the air, or into a sulphuric acid chamber, the gases and vapours are conducted for a distance of some 20 or 30 feet through a leaden tube into a receiver of the same metal, for the purpose of condensing the sulphuric acid distilled off, together with a certain amount of sulphate of silver mechanically drawn over with it. This liquid, which contains large quantities of sulphuric acid in solution, marks from 40° to 45° on Baumé's scale.

When the attack on the granulated metal is terminated, from 4 lbs. to 5 lbs. of sulphuric acid of 58°, derived from the concentration of acid liquors obtained during the preparation of sulphate of copper, are introduced into the boiler, and, after being boiled for a few minutes, the vessel is withdrawn from the fire, and its contents allowed to settle. In this way the gold falls to the bottom; the supernatant liquor is decanted off into another receiver, and diluted with water until it marks from 25° to 30° on the hydrometer.

This solution is now introduced into a large leaden cistern, heated by steam, in which are hung copper plates, by which the silver is precipitated in the form of minute granular crystals. In order to ascertain when the whole of the silver has been precipitated, a little of the liquor is filtered, and tested with a solution of common salt; when no further turbidity is caused by its introduction, it is evident that no traces of silver remain in the liquid. In some establishments the liquor is drawn off into leaden reservoirs, where it is allowed to remain in contact with copper plates during several days, in order that the last traces of silver may become precipitated.

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* When gold bullion is refined, it must be alloyed with sufficient silver to make the proportion of gold as 1 : 2½. It is then granulated and treated in the same way as silver bars containing gold.

When the argentiferous and auriferous alloy is very impure, the solution in sulphuric acid is often turbid, and gives rise to deposits in the bottoms of the vessels in which it is allowed to stand. These sediments are transferred to a separate boiler, diluted with water, and precipitated by sheets of copper. The precipitate thus obtained is then washed and dried, and, after being melted and granulated, is again treated by sulphuric acid. The clear solutions obtained in this way are subsequently transferred to the vessels in which the precipitation of silver is effected.

In some cases, instead of proceeding as above described, the muddy deposits are first washed with warm water, and then allowed to settle. The residue, which principally consists of a mixture of gold and metallic sulphates, is filtered, and then boiled in a platinum vessel with sulphuric acid; by which means a clear solution of silver is obtained, and the gold is deposited at the bottom of the vessel. This is again treated with sulphuric acid, after which it is sufficiently pure to be run into ingots, whilst the solution of silver is transferred to the precipitating tank.

When, instead of using platinum vessels, cast iron boilers are employed, the alloy to be attacked may be either in a granulated form or in the state of ingots, and the operation can be conducted on from five to ten cwt. of metal at a time. If this method be adopted, acid of a less degree of purity may be employed; and it is also found that solution is effected in cast iron vessels with a less expenditure of acid than when platinum is the material used.

The solutions are transferred in an almost boiling state from the iron vessels in which they have been prepared, to a leaden tank, which is afterwards filled up to about two-thirds its capacity with mother liquors from the crystallisation of sulphate of copper. A jet of loose steam is introduced into the tank, and the liquor boiled until it has become saturated with sulphate of silver. When this takes place, the steam is turned off, and the liquid, after standing an hour to settle, is drawn, either by a syphon or tap, placed about ten inches from the bottom, into the precipitating vat. After several successive washings, the gold is removed, again boiled with sulphuric acid in a platinum vessel, and then melted into bars.

The silver precipitated by copper plates is carefully washed and dried in an iron pan over a coke fire, or is formed into blocks by means of an hydraulic press. These must be gradually heated before being fused, as they invariably retain a considerable amount of moisture, and might cause the crucibles to crack if at once introduced into them. When sufficiently dry, the compressed silver is fused into ingots. The silver thus refined contains an inappreciable amount only of gold, but is still alloyed with about two thousandths of copper.

In most instances the gold obtained by treatment with sulphuric acid is again boiled, in the same way, in a platinum vessel with fresh acid, and then melted into bars. In some cases the alloy is fused with three times its weight of silver, treated with nitric acid, and the residue, after being carefully washed, melted into bars. The ingots obtained at refineries working as above described generally contain about 998 thousandths of gold.

The acid solution of sulphate of copper obtained by decantation from the crystals of precipitated silver, contains, in addition to the copper originally present in the alloy, that which corresponds to the amount of sulphate of silver decomposed by the copper plates. This is evaporated in leaden vessels, until it marks 40° Baumé; when, on cooling, it deposits pale blue crystals, which contain a less amount of water of crystallisation than ordinary sulphate of copper. By means of successive evaporations, the mother liquor furnishes other crops of crystals of sulphate of copper; and, finally, a dark-coloured liquid, of from 52° to 58° Baumé, is obtained, which chiefly consists of dilute sulphuric acid, and is employed in the course of succeeding operations. Sulphate of copper, which has crystallised from solutions containing a large excess of sulphuric acid, is not of a marketable quality, but requires to be dissolved in water, and re-crystallised, when it affords the usual crystals of a deep blue colour, containing five equivalents of water of crystallisation. The crystallisers are of wood, lined with lead, three feet wide, three feet deep, and four feet six inches in length. The floor of the crystallising house is of lead, and provided with gutters for carrying off, into suitable reservoirs, any liquors that may be accidentally spilt.

Some experiments made in the Lille Mint, with the view of substituting iron for copper plates in the process of refining, did not, on the whole, afford satisfactory results.

When the proportion of silver in the alloy does not amount to from 20 to 30 per cent. of the whole, instead of treating it directly by sulphuric acid, it is first granulated, and roasted in a furnace at a low red heat. The oxidised mixture is afterwards boiled in the weak acid liquors from the crystallisation of sulphate of copper, by which a portion of the copper is removed, and the residue subsequently refined in the usual way.

Assay of Gold Bullion.—The process employed for assaying gold bullion by the assayers of the Mint and the Bank of England is similar to that employed at the Paris Mint, and the weight operated on is often the same. One-half gramme is accurately weighed, and subjected to cupellation with a proper quantity of lead, and a portion of pure silver, about three times the weight of the gold supposed to be present in the alloy.

The resulting button is then flattened into a disc, about the size of a sixpence, and, after being annealed, is passed through a flatting-mill until it has been drawn into a riband from 2½ to 3 inches long, which is again annealed, and coiled into a spiral by rolling between the finger and thumb. The *cornet* is now introduced into a long-necked flask, containing about an ounce of pure nitric acid, of 22° Baumé = 1·18 sp. gr., and boiled until red fumes have ceased to be evolved. This acid is carefully poured off, and the cornet again twice boiled, for about ten minutes each time, with acid of 32° Baumé = 1·28 sp. gr. In the two last boilings, a piece of charcoal, consisting of half a charred lentil, is introduced into the flask for the purpose of preventing the ebullition from taking place irregularly and with sudden bursts, which would be liable to break the cornet and project a portion of the liquid out of the flask.

After this second attack, the cornet is washed twice with distilled water. The flask is then filled with water, and reversed, with care, into a small crucible of fine clay, into which the cornet is allowed to fall, gently and without breaking. The water which covers the cornet is now poured off, and the crucible and its contents are heated to redness in the muffle; taking care, however, not to raise the temperature sufficiently to cause the fusion of the gold. From the weight of the cornet obtained, the fineness of the alloy is calculated.

The cornet, after boiling in nitric acid, is of a spongy texture, of a brownish-yellow colour, and exceedingly fragile, so that it would be impossible, in this state, to touch it with the fingers without breaking it; and it is consequently necessary to transfer it carefully with the water contained in the flask. By heating it to redness, in the way described, its volume is reduced to about one-third its original size; it at the same time assumes a metallic lustre, and acquires a degree of cohesion which admits of its being readily handled without fear of loss.

The return is made to the Mint in decimals, the assayer's weights being so subdivided as to give him the value in thousandths of the original half gramme. The Bank return is made to the one-eighth of a carat grain "better" or "worse" than standard, and tables are employed for the conversion of assays expressed in decimals, and conversely.

The quantity of lead necessary for passing an alloy of gold on the cupel depends on the amount of copper present. The following proportions have, after numerous experiments, been adopted at the Paris Mint:—

Amount of Gold in Copper Alloy.	Amount of Lead necessary to remove the Copper by Cupellation.
1000	1 part
900	10 parts
800	16 ,,
700	22 ,,
600	24 ,,
500	26 ,,
400	34 ,,
300	34 ,,
200	34 ,,
100	34 ,,

For the assay of ordinary bar gold from California, Australia, or New Zealand, &c., in which the proportion of copper is always exceedingly small, the gold is passed to the cupel, with the necessary amount of silver and two grammes only of lead. But even when the operation of parting has been skilfully conducted, the weight of the cornet of gold will often be increased by a very slight *surcharge* occasioned by its retaining minute traces of silver.

This surcharge is most observable in cornets obtained from alloys containing a very small proportion of copper; since, in cases in which large quantities of this metal are present, and a considerable amount of lead has consequently been employed for cupellation, it is more than compensated for, by the loss of gold absorbed by the cupel. In the assay of alloys having an intermediate composition, the loss of gold is not unfrequently counterbalanced by the surcharge of silver, and the true fineness of the alloy thus directly obtained.

The following results of numerous synthetical experiments made in the Paris Mint, furnish the necessary data for the calculation of a table of corrections :—

Fineness of Gold.	Result obtained.	Differences.
900	900·25	0·25
800	800·50	0·50
700	700·00	0·00
600	600·00	0·00
500	499·50	0·50
400	399·50	0·50
300	299·50	0·50
200	199·50	0·50
100	99·50	0·50

The above experiments were made on mixtures of fine gold and pure copper, in the proportions indicated in the table.

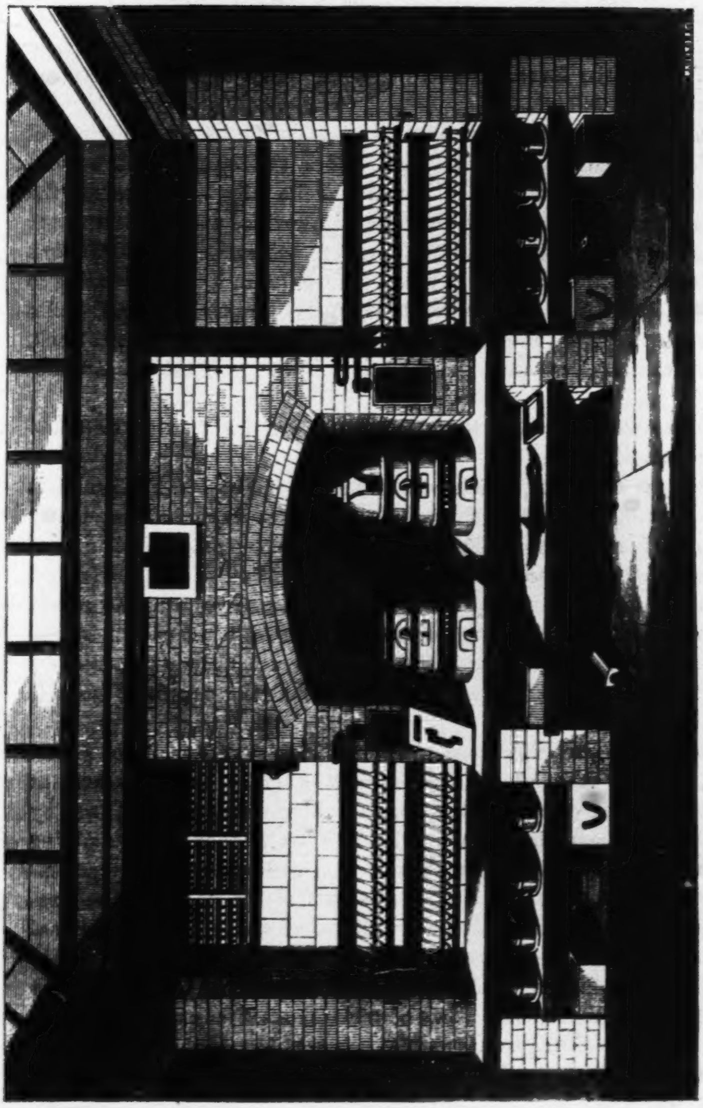
The last traces of silver may be removed from the cornet by treating it, before annealing, with fused bisulphate of potash in a porcelain crucible. When sufficiently cool, the whole is heated with water containing a little sulphuric acid, and the cornet dried and ignited. By this means gold of almost absolute purity may be obtained.

The arrangement of Mr. Claudet's laboratory, Coleman Street, London, which is probably one of the most complete for the assay of gold and silver bullion, is represented Fig. 38.

The cupellations are performed in two French cupel furnaces, made of fireclay, bound with flat iron bands, and standing in an arched recess in the wall of the laboratory, before which is an iron screen (divided in the drawing for the purpose of showing furnace) in order to prevent radiation.

Slides in this screen can be raised or lowered at pleasure, and are provided with counterpoises similar to the sash weights of a window. The tongs for handling the cupels, with its pasteboard guard to protect the hands of the operator from the heat, are seen between the furnaces, whilst a side view is obtained of another similar pair lying on the bench beneath. Under this is kept the coke to be employed as fuel, and to the left of the furnace, lying against the wall, is the pasteboard shield, with a piece of green or blue glass in it, used by the assayer to protect his eyes from injury whilst looking into the

Fig. 38.



Assay Laboratory.

muffle. The racks in the upper left-hand corner are for the storage of cupels, which, after being made in a screw press, are laid aside in shallow iron trays until nearly dry, when they are removed, together with the trays, to the racks seen, behind the tongs, on the shelf below the furnaces, where the last traces of moisture become expelled.

On either side of the furnaces are ranges of flasks for the attack of the inquartated gold, each of which rests on the top of a small button-shaped piece of metal, which has around its edge, apertures for the issue of gas jets. Each of these burners is provided with a separate tap; and a large stopcock, seen at the end of each series, is used to turn on, or cut off, the supply. A guard with holes in it, corresponding to the different burners, keeps the flasks in their respective places, whilst their open ends are inserted into holes in the tile work at the back, which communicate with the chimney and carry off the gases generated.

The vessels resembling teapots are employed for pouring acid into the parting flasks.

This gas apparatus, which was first employed by Mr. F. Claudet, is now adopted by most of the principal assayers, both in this country and on the continent of Europe, and has, among other similar establishments, been introduced into the French Mint.

In estimating the fineness of gold, it is sometimes usual to regard the whole mass as weighing 24 carats; and thus when gold is said to be 18 carats fine, it means that if the whole weight be represented by 24, the proportion of gold will be 18, or that the weight of gold is $\frac{18}{24}$ths of the whole.

Touchstone.—In order to ascertain the fineness of articles of jewellery, and of other small objects from which a sufficient amount cannot be scraped off for the purpose of an ordinary assay, trial by touchstone is sometimes employed. This is a method of ascertaining the approximate value of an alloy of gold with other metals by comparing the colour, &c. of a minute portion of the object to be examined with small bars of similar alloys, of which the proportional composition has been previously determined. These bars are called *touch needles*, and are rubbed on a smooth piece of black basalt or slate, called a touchstone. Sets of touch needles may consist of pure gold; of gold of 23½ carats; of 23 carat gold; and thus down to 20 carats, after which the difference between each needle may be a whole carat.

The marks made respectively by the several needles and the object to be tested, are afterwards wetted with a little aquafortis, and, from the effects produced, the composition of the alloy is inferred; since its fineness is supposed to correspond with that of the bar, which affords a trace on the stone similarly affected by the action of acids. In some countries where trinkets are required to be assayed in this way, a great variety of needles is necessary; but in England the touchstone is very little employed, although it can sometimes be used with advantage to indicate the quantity of silver necessary for inquartation. The British standard for gold coin is 22 carats, and the alloy employed consequently contains 916·66 parts of gold in a thousand. The standard employed in France and America for the gold currency is 900 thousandths.

SILVER.

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CHAPTER XII.

MODE OF OCCURRENCE AND GEOLOGICAL POSITION.

NATIVE SILVER—ANTIMONIAL SILVER—NATIVE AMALGAM—PRINCIPAL SILVER ORES —SULPHIDES—ARSENIDES—SELENIDES—TELLURIDES—CHLORIDE—IODIDE—BROMIDE—CARBONATE—SILICATE—CHIEF SOURCES OF SILVER—GEOLOGICAL POSITION.

Mode of Occurrence.—This metal occurs in the following forms of combination:—

NATIVE METAL AND ALLOYS.

Native Silver.—This metal sometimes occurs in a state of almost chemical purity, but it is more frequently associated with copper, gold, bismuth, antimony, or some other metal or metals. Native silver is often found in connexion with various argentiferous ores, and has sometimes been met with in masses of considerable size. The most remarkable of these have been obtained from the mines of Freiberg in Saxony, Kongsberg in Norway, and those of Huantaya in Southern Peru, from which a specimen weighing over eight cwt. was once procured. A mass of native silver from Kongsberg, in the Royal collection at Copenhagen, weighs about five cwt. It is found both crystallised and in arborescent and filiform shapes. The filiform varieties are often composed of one or more series of octahedrons, closely united, or arranged in parallel rows; this structure is apparent in many of the specimens from Norway and Mexico.

The alloys of silver and gold are exceedingly numerous, and although native gold has never been found free from silver, it is in some cases alloyed with that metal to such an extent, that the resulting compound can only be regarded as native silver containing traces of gold. Silver obtained from the treatment of ordinary argentiferous ores frequently contains gold, but generally speaking in small quantities only. In some districts, however, as at Virginia City in the State of Nevada, one-third of the value of the bullion produced, arises from the amount of gold which it contains.

Antimonial Silver.—Discrasite occurs in veins at Altwolfach in Baden, Wittichen in Suabia, Andreasberg in the Hartz, near Coquimbo, Chili, and elsewhere. Colour, silver white; composition, antimony 23, silver 77 per cent.; formula, $Ag^2 Sb$. Heated before the blowpipe, gives off fumes of antimony.

This substance does not occur in sufficient quantities to possess any great commercial value.

Bismuth Silver.—A rare alloy of silver and bismuth, with a little copper and arsenic; occurs in the mine of San Antonio, near Copiapo, Chili. It contains 60 per cent. of silver.

AMALGAM.

Native Amalgam.—Is found in the Palatinate, at Sala in Sweden, Almaden in Spain, and in various mines in Chili, &c. It is frequently crystallised, of a silver white colour, is brittle, and emits a grating sound when cut. There are two known varieties: the one is represented by the formula $Ag\ Hg^2$, and the other by $Ag\ Hg^3$. The first is composed of silver 34·8, mercury 65·2, and the second of silver 26·25, mercury 73·75 per cent.

A silver amalgam of some commercial importance is found in the mines of Arqueros in Chili, and has been hence named *Arquerite.* According to Domeyko, its composition is represented by the formula $Ag^6\ Hg$; it consists of silver 86·49, mercury 13·51 per cent.

ORES.

The ores of silver which occur in greatest abundance, and which are consequently the most important, are the following:—

Silver Glance.—Vitreous Sulphide of Silver.—This is the most important ore of silver, and contains, when pure, silver 87·04, sulphur 12·96 per cent. Its formula is $Ag\ S$. It is found in Europe in the mines of Annaberg and Joachimstahl, and other mines of the Erzgebirge; at Schemnitz and Kremnitz in Hungary, and at Freiberg. It is abundant in the mines of Peru and Mexico, and also in those of the State of Nevada.

Stephanite.—Brittle sulphide of silver is the ore of next greatest importance. This is a double sulphide of silver and antimony, containing, when pure, silver 70·4, antimony 14·0, and sulphur 15·6 per cent. Its formula is $Ag\,S + \frac{1}{6}\,Sb^2\,S^3$. It is found in nearly all the silver mines of Europe, and occurs abundantly in Mexico, Peru, and in the Comstock lode, State of Nevada; beautifully crystallised specimens have been frequently obtained from the California mine worked on that vein.

Pyrargyrite.—Ruby Silver.—An important ore in the Mexican mines, as well as of those in the Reese River district in Nevada. It is composed of the same substances as Stephanite, but in different proportions. When pure, its composition is, silver 58·98, antimony 23·46, and sulphur 17·56 per cent. Its formula is $Ag\,S + \frac{1}{3}\,Sb^2\,S^3$.

Chloride of Silver.—Horn Silver.—This ore is composed of silver 75·33, chlorine 24·67 per cent., and is represented by the formula $Ag\,Cl$. This substance is found in most of the silver mines both of Europe and America, and occurs in greatest abundance near the outcrops of the veins. It fuses in the flame of a candle, giving off acrid fumes; and if moistened and rubbed with a piece of iron or zinc, becomes externally coated with a thin film of metallic silver. With a little carbonate of soda it is readily reduced before the blowpipe, and affords a button of silver.

In addition to the foregoing, which yield the larger proportion of the total amount of silver annually produced, there are numerous other minerals containing this metal, but which, from their rarity, may be regarded rather in the light of mineralogical curiosities than as ores of silver. A large amount of silver is likewise extracted from galena, with which it is associated in the form of sulphide.

The following is nearly a complete list of the minerals formed by the combination of silver with other bodies:—

SULPHIDES, ARSENIDES, SELENIDES, AND TELLURIDES.

1. *Silver Glance.*—Sulphide of silver.
2. *Discrasite.*—Antimonial silver.
3. *Naumannite.*—Selenide of silver; occurs in the Hartz.
4. *Stromeyerite.*—Sulphide of copper and silver.
5. *Eucairite.*—Selenide of copper and silver; found in Sweden.
6. *Sternbergite.*—Sulphide of silver and iron, flexible, and marks like plumbago; found in Bohemia and Saxony.

7. *Hessite.*—Telluride of silver; found in Siberia.
8. *Miargyrite.*—A rare sulphide of silver and antimony.
9. *Pyrargyrite.*—Ruby silver.
10. *Proustite.*—Light red silver ore, composed of sulphur, arsenic, and silver; found in Bohemia, Saxony, &c.
11. *Polybasite.*—A variable combination of silver, copper, antimony, arsenic sulphur, and iron; chiefly found in the Mexican Mines.
12. *Freislebenite.*—Antimonial sulphide of lead and silver; containing twenty-two per cent. of the latter metal.
13. *Stephanite.*—Sulphide of silver and antimony.
14. *Xanthocone.*—Sulphide and arsenide of silver; occurs at the Himmelsfürst Mine in Saxony, and various other localities.

CHLORIDE, IODIDE, AND BROMIDE.

15. *Horn Silver.*—Chloride of silver.
16. *Iodyrite.*—Iodide of silver; found in Mexico, Chili, and at the Mines of Hiendelaencina, in the province of Guadalajara, Spain.
17. *Bromyrite.*—Bromide of silver; occurs in Chili, Mexico, and at Huelgoët in Brittany.
18. *Embolite.*—Chlorobromide of silver; found in Chili, Mexico, and in the State of Honduras.

CARBONATE.

19. *Selbite.*—Regarded as a carbonate of silver, but probably only a mechanical mixture.

SILICATE.

20. . . . . This combination is stated by Mr. J. Napier, Jun. to exist in the ores of the district of Reyes, Mexico.*

Few metals enter into a greater variety of natural combinations, or are found over a wider geological range than silver. It is said to exist in minute traces in some organic bodies, and in the waters of the ocean. A certain amount of this metal invariably accompanies native gold, and it would be almost as difficult to find a specimen of galena from which traces of silver could not be extracted, as to meet with native gold entirely free from it.

The whole of the silver of commerce is derived from the three following sources:—

1st. From silver ores proper, in which this metal predominates in value over those with which it is associated.

2nd. From refining the native alloys of gold and silver.

3rd. From the desilverising of lead, and the treatment of certain argentiferous copper ores.

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* *Mining and Smelting Magazine*, vol. i. p. 105.

We shall at present chiefly confine our attention to a description of the principal localities producing ores belonging to the first class, since the second source of silver has been already treated of in the portion of this volume devoted to gold, and the third will receive some attention in subsequent chapters.

The principal mining districts in Europe which have been extensively worked for ores of this description, are those of Saxony and Bohemia, some localities in Hungary and Transylvania, the neighbourhood of Kongsberg in Norway, and that of Hiendelaencina in Spain. On the continent of America, Mexico, and the Cordilleras of Southern America, furnish examples of the occurrence of vast deposits of these ores; whilst more recently the discovery of the Great Comstock and some other veins in the new State of Nevada, has been followed by the production of an amount of silver almost without parallel in modern history.

GEOLOGICAL POSITION OF SILVER.—As before remarked, the rocks enclosing deposits of ores of this class differ widely with regard to their geological age. They occur in true veins in the older crystalline and metamorphic rocks, and when so situated have often been worked to great depths without any apparent change of character or diminution of produce. The mines of Kongsberg in Norway and Freiberg in Saxony, are remarkable examples of this mode of occurrence. In South America, on the contrary, a great portion of the silver ores are found forming veins in calcareous rocks, and often running parallel to their stratification. The age of the silver-bearing rocks of Bolivia and Peru is apparently carboniferous, whilst in Chili the strata in which the ores of silver are most abundant are believed to belong to the cretaceous period.[*]

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[*] Mr. Rémond has obtained fossils in sufficient numbers from the rocks in which are situated the fampus silver mines of Chañarcillo and Tres Puntas, to fix their age as belonging to the Lower Cretaceous.—*Proceedings of the California Academy of Natural Sciences, December 3rd*, 1866.

CHAPTER XIII.

PRINCIPAL SILVER MINES OF THE OLD WORLD.

SILVER MINES OF THE UNITED KINGDOM—NORWAY—SWEDEN—TRANSYLVANIA AND THE BANAT—SAXONY AND BOHEMIA—MINES OF THE HARTZ—SILVER MINES OF THE ALPS—FRANCE—SPAIN—ALTAI MOUNTAINS—DAOURIA.

SILVER mines were extensively worked in Europe long before the discovery of the richer veins of America, and some of them still continue in operation, affording considerable annual returns of this metal.

GREAT BRITAIN AND IRELAND.—The whole of the silver produced in the United Kingdom is extracted from argentiferous lead, and consequently an account of the different districts from which it is obtained, would involve a description of the principal lead mines of the country.

It may, however, be remarked that the ores obtained from the North of England, generally speaking, contain but a small proportion of silver, usually not above 1½ oz. per ton. The ores of the Isle of Man are highly argentiferous, yielding from 50 to 60 oz. of silver per ton of lead, whilst the ores of Cardiganshire and Montgomeryshire are moderately so, containing, on an average, from 15 to 25 oz. of that metal per ton of lead. The lead ores of Denbighshire usually contain a very small amount of silver. Those of Cornwall and Devon are generally richer in silver than any others in the United Kingdom, with the exception of those from the Isle of Man. The Cornish ores contain on an average about 30 oz. of silver per ton.

The produce of silver, separated from lead ore, raised from mines in the United Kingdom, from the year 1856 to 1865, both inclusive, has been, according to Mr. R. Hunt, F.R.S., as follows:—

	Ounces.		Ounces.
1856	614,188	1861	569,530
1857	532,866	1862	686,123
1858	569,345	1863	634,004
1859	576,027	1864	641,088
1860	554,002	1865	724,856

NORWAY.—The celebrated mines of Kongsberg were discovered in 1623, and have been worked, with but comparatively little interruption, from that time up to the present. On the separation of Norway from Denmark in 1814, the present Government took the mines of the territory of Kongsberg into its own hands, and appointed a Commission for the purpose of deciding which of them should be re-opened. In addition to the Gottes Hülfe their choice fell on the Kongens and Armens mines, but, at a depth of forty fathoms from the surface, the former became so impoverished that it was abandoned, and the working of the Armens alone for some time proceeded with.

At the depth of sixty fathoms the condition of the Armens mine was considered so satisfactory that it was thought advisable to extend a level from it into the adjoining mine. This operation resulted in making important discoveries, and the two properties have been since worked as one establishment. During the early part of the eighteenth century the mines of Kongsberg were remarkably productive, but from the middle of that century up to 1832 their yield was comparatively insignificant. Since that time they have been regularly worked with uniform and satisfactory results, and have now reached the depth of 280 fathoms from the surface.

These mines are situated in gneiss and crystalline slate, of which the district for a length of a hundred miles and a breadth of fifty, is chiefly composed. The silver occurs in what are called *Fahlbands*, which consist of parallel belts of rock, of considerable length and breadth, impregnated with sulphides of iron, copper, and zinc, and sometimes also with those of lead, cobalt, and silver. The iron pyrites is often more or less decomposed, giving rise to the formation of hydrated oxide of iron, the presence of which is considered in the district an indication of silver.

The direction of the ore-bearing belts is nearly north and south; they are irregular in their dimensions, but constantly preserve a certain degree of parallelism with each other, and may be traced for an extent of several miles. The amount of ore disseminated through them is usually small, but in some places it is sufficiently concentrated to admit of being profitably worked. In the Kongsberg district, there are several of these fahlbands, parallel in strike and inclination with the gneissoid and schistose strata in which they occur, and subject to the same local structure and disturbances of stratification. The fahlbands are themselves traversed by true veins containing silver ores, and experience has shown that these are never argentiferous except where they intersect the fahlbands. From this it would appear that the impregnation of the veins is dependent on the nature of the enclosing rock, which renders it probable that their metalliferous constituents were originally derived from the fahlbands, from which their removal and subsequent concentration have been effected by chemical action.

The total produce of silver from the Kongsberg mines from 1624 to 1864,—240 years,—has been, according to official returns, as follows viz. :—

From 1624 to 1805 . . .	2,360,140 marks,*	equal to	1,332,495 lbs. troy
„ 1805 „ 1815 . . .	38,112 „	„	21,517 „ „
„ 1815 „ 1864 . . .	820,956 „	„	463,498 „ „

The average annual produce has been—

From 1815 to 1833 . . .	4,141, marks,	equal to	2,338 lbs. troy
„ 1834 „ 1838 . . .	27,423 „	„	15,483 „ „
„ 1839 „ 1843 . . .	25,454 „	„	14,361 „ „
„ 1844 „ 1848 . . .	23,464 „	„	13,247 „ „
„ 1849 „ 1853 . . .	20,552 „	„	11,603 „ „
„ 1854 „ 1858 . . .	32,862 „	„	18,553 „ „
„ 1859 „ 1863 . . .	16,091 „	„	9,084 „ „

From 1859 to 1864 the annual yield has been—

1859	20,515 marks,	equal to	11,582 lbs. troy
1860	18,139 „	„	10,241 „ „
1861	14,823 „	„	8,369 „ „
1862	13,088 „	„	7,389 „ „
1863	13,890 „	„	7,842 „ „
1864	13,046 „	„	7,365 „ „

In addition to the mines of Kongsberg, some veins in the district of Skara were formerly of considerable importance, but are now abandoned, although the fahlbands of this locality closely resemble those which have proved so productive in the adjoining district.

The produce, expenses, and net profits of the Kongsberg mines, during the last thirty-one years, have been as follows :—

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* The mark of Norway and Sweden is equivalent to 3,252 gr., or 6·775 oz. troy.

STATISTICS OF THE KONGSBERG MINES.

YEAR.	PRODUCE SOLD.	EXPENSES.	NET PROFIT.
1834	£87,558	£17,407	£70,151
1835	50,171	16,215	33,956
1836	70,244	13,585	56,659
1837	52,087	21,015	31,072
1838	58,137	21,258	36,879
1839	69,879	20,110	49,769
1840	64,236	20,497	43,739
1841	50,009	22,030	27,979
1842	42,915	19,524	23,391
1843	41,398	19,811	21,587
1844	39,462	18,117	21,345
1845	36,772	16,462	20,310
1846	37,297	12,642	24,655
1847	51,831	19,146	32,685
1848	75,788	16,268	59,520
1849	49,934	18,621	31,313
1850	47,518	16,362	31,156
1851	38,140	15,975	22,165
1852	40,770	16,905	23,865
1853	36,363	16,444	19,919
1854	51,616	17,984	33,632
1855	82,448	20,294	62,154
1856	66,922	19,613	47,309
1857	49,627	17,886	31,741
1858	84,356	20,136	64,220
1859	41,888	20,996	20,892
1860	37,157	23,420	13,737
1861	30,141	24,037	6,104
1862	26,708	20,551	6,157
1863	28,836	20,982	7,854
1864	28,090	21,171	6,919
Averages.	£50,750	£18,885	£31,704

SWEDEN.—The silver mines of Sweden were formerly of considerable value, but are at present less productive. Three mines of this metal were working in 1767, viz.: that of Hellefors in the Province of Wermland, that of Segersfors in Nericia, and that of Sala in Westmannia, the last being the only one of any importance. It is very ancient, and is said at one time to have yielded large quantities of silver, but its present annual produce is only from 2,258 to 2,823 lbs. troy. The ore is a highly argentiferous galena.

HUNGARY, TRANSYLVANIA, AND THE BANAT.—The mines of this region may be divided into four principal groups. The Schemnitz district comprehends the mines of Schemnitz, Kremnitz, Neusohl, and Schmölnitz.

Schemnitz.—Schemnitz, a royal free city of mines, is situated twenty-five leagues to the north of Buda, at a height of 1680 feet above the level of the sea, in the midst of a group of mountains covered with forest. These mountains are for the most part composed of trachytes, enclosing a formation consisting of porphyry associated with syenite, passing into granite and gneiss, with subordinate beds of slate and limestone. All the mines occur in this formation. Beudant, who compared these rocks with specimens of the metalliferous porphyry, &c., brought from South America by Humboldt, long since recognised their exact identity, not only in the most minute details of colour, structure, and composition, but also in the respective positions of the different varieties, and the same similarity of the enclosing rocks would seem to exist with regard to the recently-discovered silver districts in North America.

The metalliferous rocks of Schemnitz occupy a tract of some miles in extent, and are traversed by a group consisting of five principal veins running north-east and south-west, in addition to which there are numerous others of less importance on the north side of the Paradise mountain. The largest of these, the "Spitaler gangue," has occasionally a width of sixty feet, and may be traced for upwards of four miles in length. These lodes have seldom any distinct walls, but the enclosing porphyry is frequently more or less decomposed and impregnated with iron pyrites in the vicinity of the planes of contact. The veins are rarely subject to interruption or dislocation by slides or cross-courses. The veinstone consists of fragments of the enclosing rocks (which are often decomposed to the consistency of clay), cavernous quartz, ferruginous limestone, and sulphate of baryta, with which are associated blende, iron and copper pyrites, argentiferous galena, sulphide of silver, ruby silver, and metallic silver, in addition to a small quantity of gold which is rarely visible. Sulphide of silver and argentiferous galena are the two most important minerals, and sometimes occur in distinct patches, whilst at others they are intimately mixed and disseminated throughout the gangue. The gold and silver are usually found to accompany each other, but they do not now occur in such large quantities as formerly, since the proportion of galena present in the veins evidently increases in depth.

The argentiferous ores from Schemnitz are smelted near the mines, the resulting work lead being sent to refining establishments at Kremnitz, Neusohl, and Scharnowitz, where the silver ores obtained from the other mines in the neighbourhood are also treated. The mines of Schemnitz were first opened more than eight hundred years ago, and have been worked to a depth of over two hundred fathoms ; the works have been generally conducted with great care and skill, but from the falling off of the produce of the ores, they are not now in so prosperous a condition as they were a century or more since.

The Empress Maria Theresa founded a School of Mines at Schemnitz in 1760, which was for many years celebrated throughout Europe, but since the year 1848 the number of students has considerably declined, and the rival schools of Przibram and Gratz have proportionately increased in importance.

Kremnitz is situated about five leagues north-west of Schemnitz, in a valley bounded on the right by a range of hills composed of rocks similar in their character to those constituting the metalliferous formation in the vicinity of the latter town. Enclosed in these rocks are veins closely analogous to those of Schemnitz, except that the gangue contains a larger proportion of quartz, and that the ores are generally more uniformly auriferous. The metalliferous area is here of very moderate extent, and is surrounded by a trachytic formation, which geologically belongs to a more recent epoch.

Kremnitz is one of the most ancient royal free cities of mines in Hungary, and has a mint where all the bullion obtained is sent to be parted and refined.

Neusohl lies about six leagues north-east from Schemnitz, on the banks of the river Gran, and was originally founded by a colony of Saxon miners. The mines are here situated in grauwacke, covered by limestone, and afford argentiferous copper ores. They have been worked since the thirteenth century, and annually produce about 105 tons of copper and 1,053 lbs. troy of silver.

The mines of Lower Hungary afford employment to 15,000 workmen, and yield metals of the total annual value of 360,000*l.*

In the neighbourhood of Schmölnitz are copper and other mines belonging to private individuals, which produce, in addition, a certain amount of silver and mercury. They are chiefly worked by a Company called the Waldbürgerschaft, which produces annually 850 tons of copper, 3,535 lbs. troy of silver, and about a hundred flasks of mercury.

Nagybanya.—The veins of this group lie in a chain of mountains which, proceeding from the frontiers of Buckowina, where it unites with the Carpathians, finally disappears among the sandstones on the northern frontier of Transylvania. This district produces some gold, but its annual yield of silver is small.

Abrudbanya.—Nearly all the mines of this district occur in the mountains which rise in the western part of Transylvania, between the Lapos and Maros, in the vicinity of Abrudbanya. The most productive veins are principally found in porphyries analogous to those of Schemnitz, although some of them are enclosed in mica slate grauwacke, and limestone. The principal veins are those of Nagyag, Körösbanya, Offenbanya, Vöröspatak, Boitza, Csertesch, Fatzbay, Füzes, Vulkoj, Porkura, Butschum, and Toplitza. The mines of all these localities produce gold, together with a certain amount of silver, which is smelted at the works at Zalathna. They are celebrated for producing ores of tellurium.

Banat of Temeswar.—The mines of the Banat chiefly occur in the mountains that enclose the valley of the Danube at Orsova, through a narrow gorge in which the river finds a passage. The most important mines of this district are those of Oravitza, Moldawa, Szaszka, and Dognaczka. They principally afford argentiferous copper ores, yielding about 120 oz. of silver to the ton, together with a little gold.

These ores are obtained from veins and deposits chiefly occurring between the mica slate and limestone, or between syenite and limestone. Well defined veins are also found enclosed in syenite and mica slate.

The mines of Hungary, Transylvania, and the Banat, produce about 92,000 lbs. troy of silver annually.

Saxony and Bohemia.—The chain of the Erzgebirge, which, on the western bank of the Elbe, separates Saxony from Bohemia, has for many centuries produced a certain annual amount of silver. The silver mines of this district have been opened on veins which are, for the most part, enclosed in gneissoid rocks; their thickness rarely exceeds a few feet, and they are associated in groups, whose relative importance has varied considerably at different periods. Mining operations are said to have been first commenced in this district as far back as the beginning of the tenth century, but there are no authentic records on this subject dating prior to about the close of the twelfth, when the still productive mines of Freiberg began to be worked. The mines of Saxony suffered severely from the effects of the Thirty Years' War, during which, many considerable mining towns were entirely destroyed, and some of them have never recovered their former importance and prosperity. Freiberg, Marienberg, Annaberg, Schneeberg, Johann-Georgenstadt, and Schwarzenberg in Saxony, and Joachimsthal in Bohemia, have at various periods been the chief local centres of this mining region. More than nine hundred different veins are said to exist in the Erzgebirge district, which, in accordance with the nature of their gangues, have been divided by Von Weissenbach into the four following classes :[*]—

1. Quartz veins, containing iron pyrites, mispickel, blende, and galena; and affording silver ores of a moderate percentage.

2. Brown spar veins, yielding the same ores as the preceding, but richer in silver.

3. Veins of which the gangues are composed of carbonate and oxide of iron, fluor spar, and sulphate of baryta; but not so metalliferous as the two preceding. These sometimes extend into the Zechstein formation.

4. Veins with calcareous gangues, occasionally affording rich ores.

The characteristics of the different classes of veins do not, however, appear to be so marked as to admit of their being divided into any definite number of well-defined systems.

Freiberg.—The Freiberg mines afford an interesting example of silver veins retaining their character and richness at considerable depths. Many of the mines now exceed 230 fathoms in depth, and notwithstanding the increasing expenditure incident to this circumstance, their prosperity has continued to be constantly progressive. The most celebrated and productive of the Freiberg mines during the present century, have been those of Himmelsfürst, Himmelfahrt, and Beschertglück; and with a view to facilitate their drainage, it has been proposed to bring up a deep adit from the valley of the Elbe, a distance of about eighteen miles.

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[*] Annales des Mines, S. 4, t. xi. p. 27.

The ores of the Freiberg mines consist of the various sulphides, particularly vitreous silver, pyrargyrite, light red silver ore, stephanite, freislebenite, and native silver. The latter sometimes occurs in large masses, and the veins occasionally yield considerable amounts of argentiferous galena. The total yield of silver from the mines of Freiberg from 1524 to the end of 1850, was 5,613,228 lbs. troy. As a general rule, the productiveness of these mines has, subject to temporary fluctuations, increased with their progressive depths. At the time of the publication of the work of Heron de Villefosse, "*Sur la Richesse Minérale*" (1819), the Himmelsfürst was the most productive mine, but since 1830 its yield has declined, and the Himmelfahrt has occupied the first position, having afforded, from that time up to 1850, 146,869 lbs. troy of silver.

Marienberg.—Formerly the mines of Marienberg, a small town twenty-five miles south-west of Freiberg, were exceedingly flourishing, and in the sixteenth century ores were frequently found there producing 85 per cent. of silver. The troubles attending the Thirty Years' War put an end, however, to their prosperity: since that period they have never exhibited any degree of activity, and their annual returns are now very inconsiderable.

Annaberg.—The production of the Annaberg district during one hundred and ninety-one years, from 1654 to 1845, amounted in value to about 850,000*l.* but the yield of the veins has decreased in depth, and their annual returns are now exceedingly small.

Schneeberg.—The Saint-George mines, near Schneeberg, were opened in the fifteenth century, in search of ironstone, but soon became celebrated for their large production of silver. Towards the close of that century a mass of ore was discovered in them which afforded nearly twenty tons of that metal; but after this period they became rapidly impoverished as silver mines, but have been worked during the last two hundred years for ores of cobalt, for which they are celebrated, and which are found in the veins formerly yielding such large quantities of silver. Mining operations at Schneeberg are now carried on only on an exceedingly limited scale.

Johann-Georgenstadt.—There are several small mines in the neighbourhood of this town, producing a little silver, but the aggregate amount is exceedingly small. The veins in this district yield a certain amount of bismuth and uranium, and some of them are worked for the ores of these metals. The mines of Schwarzenberg are no longer of any importance.

The average richness of the silver ores throughout Saxony is from sixty to seventy ounces per ton, and the annual production of the country about 80,000 lbs. troy, of which the mines of Freiberg alone yield about ninety-seven per cent.

Joachimsthal.—These mines, which are of great antiquity, and were formerly of much importance, have been worked to a depth of 325 fathoms, but their produce has for many years been gradually declining in amount, and their annual yield is at present insignificant.

MINES OF THE HARTZ.—The most productive silver mines of the Hartz are those in the neighbourhood of Andreasberg, of which the most extensive are the Samson and Neufang mines, which are worked to a depth of 430 English fathoms. They were discovered in 1520, and produce argentiferous galena, in addition to silver ores properly so called. The district yielding the largest amount of argentiferous galena is that of Clausthal, which comprehends several mines worked at a depth of above 300 fathoms. The most extensive of these are the Dorothea and Carolina, the latter of which furnishes a large proportion of the annual produce of the district. The grant of the Dorothea mine extends over a length of 257 yards on the run of the vein, with a moderate width at right angles to its direction. Within this small area there were extracted, from 1700 to 1807 inclusively, 553,200 lbs. troy of silver, besides 38,441 tons of lead, and 1,192 tons of copper. The profits realised from this mine, together with that of Carolina, during the same period, amounted to 1,120,000*l.*, besides which the shareholders contributed largely, by means of loans without interest, to carry on the exploration of various less productive mines in the district.

The annual production of silver from the Hartz mines is about 27,540 lbs. troy.

MINES OF THE ALPS.—The mine of La Gardette in the Oisans, Department of the Isère, was worked during the latter portion of the eighteenth century, on a quartz vein containing small quantities of gold and silver, but it never paid the expenses of extraction, and has long since been abandoned. The mine of Chalanches or Allemont, in the same Department, was regularly worked from 1768 to 1815, but is now also abandoned. The ore consists of various minerals, containing silver, disseminated in a clay which fills the fissures and irregular cavities in talcose and hornblendic rocks. This mine yielded annually, towards the close of the last century, 1,252 lbs. troy.

The mines of Kitzbühel and Röhrerbüchel in the Tyrol were formerly worked on a considerable scale, and in the middle of the eighteenth century were considered among the deepest in Europe, but were shortly afterwards abandoned. They produced copper pyrites, and argentiferous fahlerz, occurring in a clay slate. The ores from these mines, together with the products from some smaller workings in the neighbourhood, were smelted at the works of Brixlegg, near Schwatz. The produce of the mines of the Tyrol, towards the middle of the last century, amounted annually to 6,296 lbs. troy, and was before that time considerably greater, but has since declined in importance. There are mines of argentiferous copper ore at Schladming, Feistritz, Walchern, and Kallwang, but their annual product of silver is unimportant. The mines of l'Argentière, Hautes Alpes, were probably worked by the Romans, but are known to have been in active operation in the twelfth century. The ores contain about 50 per cent. lead, and 0·0024 of silver. These mines are worked on three veins, of which the largest is 16 feet in width. Although not belonging to the above category, it may be mentioned that the mines of Wittichen in the Black Forest produced, some years since, 880 lbs. troy of silver per annum.

NORTHERN AND CENTRAL FRANCE, &c.—Huelgoët in Brittany, although strictly speaking a mine of argentiferous galena, produces an ochreous substance containing about 30 oz. of silver per ton, in the form of *chloride*, which was until recently extracted by a process of amalgamation somewhat similar to that formerly employed at Freiberg.

The mines of Pontgibaud produce large quantities of argentiferous galena, rich in silver; these are treated by a process of smelting which will be described with considerable detail in a subsequent chapter. In addition to the mines of Pontgibaud, Puy-de-Dôme; of Poullaouen, and Huelgoët in the Department of Finistère; of Vialas in the Lozère; and of Pontpéan, Department of Ille-et-Vilaine, there are in France various other localities affording argentiferous lead ores; but as these cannot be strictly regarded as ores of silver, a detailed description of them does not come within the limits which we have assigned ourselves.

SPAIN.—The silver mines of Gaudalcanal and Cazalla, to the north of Seville, occur in mica slate, and were formerly highly productive, but are now of little importance; there were also mines at Villa-Guttier, not far from that city, which are said to have furnished 170 marks of silver daily during some portion of the seventeenth century. A silver vein was also formerly worked in the Sierra Almagrera, and was followed to a depth of above a hundred fathoms. The most important silver mines of Spain are, however, those of Hiendelaencina.

The mines of this district are situated about 70 miles north of Madrid, in the province of Guadalajara. They were discovered in 1843 by a peasant who had been in Mexico, and who on his return to his native country, remarked the resemblance which a large stone situated by the side of the road, near the village of Hiendelaencina, had to the ores he had assisted to work in the Mexican mines. On examination, by the celebrated chemist Orfila, this stone was found to be an exceedingly rich ore of silver, and proved to be part of a lode which outcropped at that point.

Shortly after this discovery a concession was obtained, and the first of the numerous mines of this now widely-known district was commenced under the name of The Santa Cecilia, in honour of the patron saint of the village. Of the other mines which were subsequently opened, the most remarkable are the Suerte, Fortuna, Verdad de los Artistas, Relampago, San Carlos, and Vascongada, all of which have yielded immense wealth. There are numerous other lodes in the district, but, up to the present time, that on which these mines are situated has alone been worked with advantage. Its direction is east and west, with an underlie south, and its average thickness scarcely two feet; the enclosing rock is gneiss, accompanied by a coarse talcose slate. The vein is chiefly composed of sulphate of baryta, together with a little quartz; carbonate of iron is, however, often found associated with them, especially in the richer parts of the lode.

Many varieties of silver ores have been found in this district, but by far the most abundant is silver glance; beautiful specimens of ruby silver are also frequently met with. The average yield of the ores produced is about 4½ oz. of silver per quintal, or 90 oz. per ton.

The greater portion of these ores has been purchased by the Bella Raquel Company, and reduced at their establishments at Constante, about three miles from the mines, to which there is an excellent cart road. A portion of the richest ore was, however, at one time sent to England for treatment, but the whole produce of the mines is now again purchased by the Bella Raquel Company. These mines have declined in their yield since 1858, many of them being at the present time far less productive than formerly. The mines of Hiendelaencina have been worked to a depth of about 200 fathoms, and the concessions are generally 200 yards in length by 100 yards wide.

The production of these mines since 1846, when they were first extensively opened, has been as follows:*—

YEAR.	FINE SILVER.				
	marks	oz.	ochs.	Spanish ounces.	
1847	4,958	1	6	39,665·7	7,578,536·20 English ounces.
1848	17,345	6	4	138,766·5	
1849	21,633	6	4	173,070·5	
1850	48,516	6	6	388,134·8	
1851	68,029	7	2	544,239·3	
1852	64,616	4	2	516,932·3	
1853	69,178	1	5	553,425·6	
1854	89,784	3	4	718,275·5	
1855	80,359	3	4	642,875·5	
1856	67,730	2	4	541,842·5	
1857	65,072	0	0	520,576·0	
1858	86,052	1	5	688,417·7	
1859	67,412	2	4	539,298·5	
1860	50,639	6	3	405,118·3	
1861	40,780	3	5	326,243·6	
1862	31,745	2	3	253,962·4	
1863	24,598	2	3	196,786·4	
1864	19,019	0	3	152,152·4	
1865	9,362	5	4	74,901·5	
1866 June 30, 6 months	5,226	1	1	41,809·1	
Silver in Ore sent to England, as per Assay. .				740,210·2	
Total Produce, 19½ years.				8,196,704·3	

ALTAI MOUNTAINS.—The most important silver mine of this district is that of Zméof, situated to the north-west of the high mountains in 51° 9′ N. lat., and 79° 49′ long., east of Paris. It is worked on a vein containing chloride and sulphide of silver, native silver and native gold, together with various ores of copper, generally more or less argentiferous. The gangue consists of sulphate of baryta, carbonate of lime, quartz, and occasionally, fluor spar. The principal vein, which is very large, has been traced for a distance of several hundred fathoms, and is worked to a depth of 95 fathoms. This vein has near the surface a considerable inclination, but becomes nearly vertical in depth. Its roof is of clay slate, but on the foot wall the slate is much mixed with hornstone. It throws out numerous branches, is not unfrequently intersected by cross-courses, and was more productive near the surface than at the present time.

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* Communicated by Messrs. Taylor and Sons.

The other silver mines of this region are those of Tcherepanofsk, three leagues south-east of Zméof; those of Semenofsk, ten leagues south-east; those of Nicolaiefsk, twenty leagues to the south-west; and of Philipofsk, ninety leagues north-east of the same locality. The last mine lies on the extreme frontier of Chinese Tartary.

DAOURIA.—The name of Daouria is given to the great mountainous tract extending from Lake Baikal to the Eastern Ocean. The metalliferous portions of this district consist of granite, hornblendic shales, and slates, overlaid by a grey limestone, sometimes silicious and argillaceous, in which are veins of argentiferous lead ores. The silver from this district appears to be chiefly, though not entirely, derived from galena, and is estimated to amount to about 21,000 lbs. troy annually. This silver contains a small proportion of gold. Many of the mines of Daouria are reported to be, at the present time, nearly exhausted.

CHAPTER XIV.

SILVER MINES OF NORTH AMERICA.

PRINCIPAL MINING DISTRICTS OF MEXICO—PRODUCTION OF SILVER IN MEXICO UP TO 1845—PRODUCE OF ORES TREATED—GUANAXUATO—ZACATECAS—FRESNILLO—REAL DEL MONTE—HISTORY AND OPERATIONS OF THE REAL DEL MONTE MINING COMPANY—PRINCIPAL MINING DISTRICTS OF NEVADA—COMSTOCK VEIN—DISCOVERY OF SILVER AT VIRGINIA—YIELD DURING FOUR YEARS—SUTRO TUNNEL—REESE RIVER DISTRICT—SYSTEMS OF VEINS—AMADOR DISTRICT—RAVENSWOOD—CORTEZ—WASHINGTON—UNION—SMOKY VALLEY—TWIN RIVER—SAN ANTONIO—OTHER MINING DISTRICTS—TABLE SHOWING NUMBER OF REDUCTION WORKS, &c.

THE principal silver mines of North America are those of Mexico, although large quantities of this metal have of late years been afforded by the State of Nevada. In addition to Nevada, other portions of the possessions of the United States, and particularly Idaho and Colorado, are known to contain valuable mines of this metal; but they have been, as yet, for the most part, imperfectly explored, and it is extremely difficult to obtain reliable data from these comparatively new and outlying countries.

MEXICO.—Our information relative to the mines of Mexico is chiefly derived from the works of European authors, who have either visited the country for scientific purposes, or who, for a time, lived there in connexion with some of the different mining and metallurgical enterprises, which have at various periods been established on that portion of the American Continent by the capitalists of the Old World. By this means, although a large amount of information has been accumulated with regard to the quantities of silver obtained from the different districts, and connected with the various processes employed, both for the extraction and reduction of the ores, we are, nevertheless, without such exact geological data as would enable us to become thoroughly acquainted with the characteristics of the mining regions, and the precise nature of the relations and differences existing between the principal metalliferous centres.

The workings have, for the most part, been excavated on true veins; and deposits in the form of beds or interfoliated masses, running parallel with the stratification of the enclosing rocks, are by no means of frequent occurrence. The veins are chiefly enclosed in what have been designated, by those describing them, as primitive and transition rocks, but of which, a careful and minute examination would doubtless enable the geologist of the present day, not only to determine the relative ages, but also to assign their true position in the geological series. Humboldt calls the limestones of the district of Tasco and Catorce "Alpine" and "Jurassic limestone," whilst Burkart speaks of them as "mountain limestone;" from which it may be inferred that at least a portion of them belongs to the carboniferous period.

The summits of the highest mountains generally consist of granite, gneiss, and mica slate, but their flanks are in most cases covered by deposits of porphyritic and trappean rocks, through which the granite seldom makes its appearance. The most metalliferous rocks of Mexico are varieties of syenite and porphyry, but Humboldt considers that the determination of their relative geological ages would be an exceedingly difficult problem. They are, however, generally characterised by the presence of hornblende, and the absence of quartz. The veins of this country have generally a direction north of west and south of east, with a considerable dip, which is more frequently towards the south than towards the north, and they generally intersect the enclosing rocks at a very decided angle. The largest vein which has been worked in Mexico is the Veta Madre of Guanaxuato, which is sometimes two hundred feet in width, and which has been opened on at various points over a distance of more than three leagues. In 1845 the workings on this lode had already attained a depth of above three hundred fathoms. The lode next in magnitude is the Veta Grande of Zacatecas, which is sometimes seventy-five feet in width; although the generality of the argentiferous veins of Mexico do not possess these extraordinary dimensions, but vary in width from a few inches to six or eight feet.

The veinstone is generally quartz, and the outcrops or *crestones* can frequently be traced for long distances on the surface, standing above the level of the enclosing rocks. The ores consist of the various simple and compound sulphides of silver, which have, near the surface, frequently become decomposed, and given rise to the formation of native silver, and various secondary combinations, of which the chloride is the most important. The outcrops and shallow portions of the veins are generally drusy, and much stained by oxide of iron; and from their red colour are called *colorados* by the Mexicans, corresponding to the gossans of the Cornish miner. Below the point at which this decomposition has taken place, and consequently where the sulphides remain in their original state, the ores are called *negros* (black ores), which, according to Duport, afford seven-eighths of the total produce of Mexico.

At the date of the publication of Humboldt's "*Essai Politique*," the richest mines of Mexico, arranged in accordance with the importance of their several yields, followed each other in the annexed order.*

Guanaxuato . .	Intendancy of	Guanaxuato
Catorce . . .	„ „	San Luis Potosi
Zacatecas . . .	„ „	Zacatecas
Real del Monte .	„ „	Mexico
Bolaños . . .	„ „	Guadalajara
Guarisamey . .	„ „	Durango
Sombrerete . .	„ „	Zacatecas
Tasco	„ „	Mexico
Batopilas . . .	„ „	Durango
Zimapan . . .	„ „	Mexico
Fresnillo . . .	„ „	Zacatecas
Ramos	„ „	San Luis Potosi
Parral	„ „	Durango

The veins of Tasco, Tlalpujahua, Sultepec, Moran, Pachuca, and Real del Monte, as well as those of Sombrerete, Bolaños, and Batopilas, have, from time to time, afforded immense wealth, but their produce has been generally of a less uniform character than that of the mines of Guanaxuato, Zacatecas, and Catorce. The silver extracted from the mines of Mexico, from January 1, 1785, to December 31, 1789, is stated by Humboldt to have been as follows:—

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* With regard to the production of the precious metals, McCulloch remarks: "The silver and gold mines of Mexico have always been deemed the main source of its wealth, and unquestionably its mineral riches far exceed those of any part of America, except perhaps Peru. Before the War of Independence there were in the thirty-seven mining districts of New Spain somewhat more than 3,000 mines, producing annually about \$21,000,000 in silver, and about \$2,000,000 in gold."

Name of Provincial Treasury.	Silver extracted by amalgamation.	Silver extracted by smelting.
	marks.	marks.
Mexico	950,185	104,835
Zacatecas	1,031,360	173,631
Guanaxuato	1,937,895	531,138
San Luis Potosi . . .	1,491,058	24,465
Durango.	536,272	386,081
Guadalajara	405,357	103,655
Bolaños	336,355	27,614
Sombrerete	136,395	184,205
Zimapan	1,215	247,002
Pachuca	269,536	185,500
Rosario	477,134	191,368
Totals . . .	7,572,762	2,159,494

The above is equal to a returned production of 9,732,256 marks, or about 6,100,000 lbs. troy. Humboldt, however, estimates that silver to at least one-fifth of the above amount, had not passed through the Government Offices; and consequently the true produce would thus become 11,678,659 marks, or 7,314,344 lbs. troy.

The mines of Zacatecas were first opened in 1548, and those of Guanaxuato ten years later, about which time the patio process of amalgamation was also introduced. The annual total yield of the Mexican mines at this period has been estimated by Humboldt at from $2,000,000 to $3,000,000, and it subsequently increased, during the eighteenth century, until it finally reached $23,000,000. The production appears to have reached its highest point between 1800 and 1810, when the average coinage of gold and silver at the various mints in Mexico was $23,664,622, the ratio of the former metal to the latter being, by weight, 0·0029 : 1; and in value, 0·05 : 1. During the War of Independence a great falling off took place in the production of the precious metals, which between 1810 and 1845 did not average above $12,000,000, for silver, with a little over $100,000 in gold. Since 1850, however, the mines of Mexico have regained their ancient prosperity, and their present annual produce cannot be much less than $26,000,000 in silver, and $3,200,000 in gold. It is, however, to be regretted that our information with regard to the production of the Mexican mines does not come down to a recent date; since no investigations on this subject appear to have been made, by any competent authority, since the appearance, more than twenty years since, of the works of Duport and Chevalier. The latter author estimates the total produce of silver from the Mexican mines from the earliest period up to 1845, at 162,858,700 lbs. troy.

The quantity of silver obtained from the ores by amalgamation was, at the close of the last century, in the proportion of 3·5 : 1 of that produced by smelting. This ratio was ascertained from the general table prepared at the provincial treasuries of the different mining districts. In some of the districts, however, as for instance, those of Sombrerete and Zimapan, the produce from smelting was greater than that from amalgamation. In 1846 it was, however, estimated by Mr. J. Phillips, that one-eighth only of the silver produced in Mexico was at that time obtained by smelting.* The result of investigations made some years since, showed that the average richness of all the ores treated from the Mexican mines was from 2½ to 3 oz. per quintal, or that the 1,878,400 lbs. troy of silver, produced from the country in prosperous years, were extracted from 446,428 tons of mineral, of which a portion was smelted and the remainder amalgamated. This would give a mean average produce of a little more than 50 oz. per ton.

In the district of Pachuca, the ores of the Biscaina vein were, in 1803, divided into three classes. The minerals of the first class contained, on an average, about 110 oz. per ton, those of the second 45 oz. and the poorest only 30 oz. per ton. In the district of Tasco, the ores of Tehuilotepec afforded, on an average, about 40 oz. of silver per ton. In 1791 the Valenciana mine of Guanaxuato, then in its most flourishing condition, yielded ores of different degrees of richness, in nearly the following proportions :—

In 1,000 parts.	Yield of silver per ton.
5	3,630 oz.
8	1,500 „
152	560 „
815	60 „

In the above year the total amount of silver extracted from the mine was 360,000 lbs. troy, of which the ores affording 50 oz. of silver per ton represented 123,508 lbs.

Space will not allow us to give even a brief description of all the productive mining regions of Mexico, and we shall consequently confine ourselves to certain details relative to some of the best known and most important districts of that country, at the same time selecting such as may serve as types of silver mining generally, on this portion of the American continent. About the year 1821, it having become evident to the Mexican Government that the pecuniary resources of the country were not sufficient to admit of the efficient development of its mines, a proposition was made to allow foreigners to participate with the natives in mining undertakings, and thus to allow of the introduction of foreign capital. This proposition was eagerly responded to by European capitalists, and in 1829 seven large English Companies, one German, and two American Companies were in operation. The English Companies were the Real del Monte, the Bolaños, the Tlalpujahua, the Anglo-Mexican, the United Mexican, the Mexican, and the Catorce. Of these the United Mexican and the Real del Monte are now alone in operation.

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* "Descriptive Notice of the Silver Mines and Amalgamation Process of Mexico." By John Phillips. London, 1846.

Guanaxuato.—The Veta Madre traverses at the surface a conglomerate possessing many of the characteristics of a red sandstone, and subsequently enters a foliated rock of greenish colour, sometimes talcose, enclosing beds of serpentine, occasionally including syenite and a bluish clay slate. The inclination of the vein is about 45° southwest, and its width is seldom less than thirty feet, but it sometimes increases to two hundred. It is generally divided into three bands, separated by intervening sterile rock. The central band is the widest and most productive, but the upper division has also been much worked, and has produced large quantities of ore, whilst the lower branch is, generally speaking, less rich than the other two. This lode, although partially explored over a length of more than three leagues, produced the chief portion of its silver within a space of about 1,000 fathoms, and particularly from that part of the vein comprehended within the concessions of Valenciana, La Cata, Mellado, and Rayas. The Veta Madre is not, however, the only silver vein which has been successfully worked in the neighbourhood of Guanaxuato, as there are others situated a few leagues to the north, which have been opened in the concession of the Pavillon and La Luz. The first of these yields ores containing a large proportion of ruby silver, whilst the veinstone of the latter is much stained by blue carbonate of copper.

The vein worked in the mine of the Asuncion, which is about a league to the north of the Veta Madre, and runs parallel with it, is, like the principal vein of the district, divided into three bands, has the same inclination, and affords ores of a very similar character. The gangue of the Veta Madre is composed of a very white quartz, containing numerous reniform cavernous concretions. The ores consist of native silver, black antimonial sulphide of silver, and small quantities of red silver ore, rarely associated with iron pyrites or galena, but occasionally with very small quantities of blende and mispickel. Gold also occurs in a very finely-divided state, and is consequently seldom visible, although grains of considerable size are sometimes met with, disseminated in the quartz. The ores of Guanaxuato generally contain less than three per cent. of metallic ingredients intermixed with the quartzose gangue.

According to Duport the average assay produce for silver, is from 0·0015 = 49 oz. to 0·0020 = 65 oz. per ton, and when the assay was below 0·0009, or 29 oz. per ton, the ores could, in 1843, be no longer extracted with advantage.* Ores containing more than 0·003 = 98 oz. per ton, are rarely met with, in any considerable quantities. The proportion of gold by weight is generally about 0·005 of that of silver, but this is subject to slight variations in different parts of the vein.

Little is known with respect to the produce of Guanaxuato previous to 1760, when the mine of Valenciana was first opened; but even at that time some of the other workings had attained considerable importance, since Gamboa speaks of the large gallery at Rayas as being a remarkable undertaking, and states that it was of such dimensions as to allow of mules entering the interior of the mine for the purpose of receiving their burdens.† In 1803, the workings in the Valenciana had already exceeded 200 fathoms in depth, which, as Duport remarks, is below the point at which the mines of Mexico usually begin to become impoverished. This gradual diminution in the average produce of a vein is called by the Mexican miners the *enborrascado*, and begins to take place in many instances even at less considerable depths. At the Valenciana, however, the great width of the lode, and the expectation of fresh discoveries, encouraged the proprietors to continue their operation to a depth of 350 fathoms, and efforts were made to compensate for the falling off in the produce of the ores, by the increase of the quantities treated. In this way the enterprise struggled forward until 1810, when the breaking out of the War of Independence caused the suspension of all operations at the Valenciana and many other Mexican mines.

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* "De la Production des Métaux Précieux au Mexique," par St. Clair Duport. P. 212. Paris, 1843.

† Gamboa, "Comentarios a las Ordenanzas," cap. 19-20.

In 1822 the re-opening of the Valenciana mine was undertaken by the Anglo-Mexican Company, who agreed with the proprietors to furnish the necessary working capital on condition of receiving one-half the net profits accruing from the operations. This Company erected at a great expense powerful steam-engines for the drainage of the mines, but, on getting out the water, it was found that the ores at the lower levels were not sufficiently rich to repay the expenses of extraction and reduction, and the deeper portions of the vein were consequently abandoned, and operations entirely confined to the extraction of ores which remained standing nearer the surface. The working of the property was for some time continued in this way, but the results obtained were so far from satisfactory that the English proprietary finally resolved to sacrifice their expenditure and abandon the enterprise to the original proprietors. The operations which have been carried on since that period have afforded but very moderate returns for the capital employed, having been sometimes barely sufficient to cover the current expenses of the undertaking. The Valenciana is the deepest mine in Mexico, and the non-success of the Anglo-Mexican Company has consequently tended somewhat to discourage further attempts at deep mining in the country.

The extraction of the water and ore from the Mexican mines is usually effected by means of *malacates*, or horse whims, around which a rope is so wound that whilst one sack of raw hide is descending the shaft another is being raised to the surface. The Valenciana mine has several shafts, but the principal one, or *tiro general*, which is 734 varas,* or 343 fathoms in depth, has a diameter of nearly thirty feet, and is one of the most remarkable engineering works in Mexico.

Next to the Valenciana, the mine of Rayas is the most important. The main shaft is the largest in the country, and was made to rival that of Valenciana. It is in the form of an octagon thirty-one feet in diameter, and secured by masonry for a depth of about thirty varas from the surface. Its total depth is 465 varas, or 215 fathoms; but its diameter is smaller downward than towards the surface. Some of the best constructed malacates in Mexico have been erected at the mine of Rayas, and at the time this neighbourhood was visited by Mr. J. Phillips (1840), each of them was worked by twelve horses driven round at a fast trot, and drew the *bota*, which when full of water weighed 1,500 lbs., from a depth of 465 varas to the surface, in eight minutes.

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* The Vara equals 33·384 inches.

In the district of Zacatecas and Real del Monte the veins are very numerous and cross each other in various places, although generally at the same angle, whilst at Guanaxuato the riches are to a greater extent concentrated in one enormous lode. The large size of this vein, and the impossibility of thoroughly trying the ground without having recourse to numerous cross-cuts, gave rise at Guanaxuato to a method of working very much resembling the tribute system of Cornwall. The miners who work on this system, which has been attended with the best results, and has led to the discovery of some of the richest deposits of ore, are called *buscones*. In this way the men, to a great extent, work the mines at their own risk, and by following up such indications as may appear to them favourable, they often meet with valuable courses of ore; whilst, on the other hand, they sometimes work for many months without gaining more than sufficient to provide themselves with the means of bare subsistence. The *buscon* receives a certain proportion, sometimes one-half the produce of the ore which he breaks, and should he fall in with a rich deposit his gains will necessarily be very large, instances not being wanting of a man making in this way from a thousand to fifteen hundred dollars per month.

In cases of this kind, however, it is optional with the owners of the mine to take the discovery out of the hands of the miners after giving them a certain notice; when this is done, each man is paid a dollar per diem without allowing him any portion of the ore. A very large proportion of the ores at Guanaxuato are, however, raised by buscones, who divide the produce of their labour with the proprietors of the concessions. The ore, after being broken and separated as much as possible from sterile matter, by hand picking, underground, is usually put into botas of bullocks' hide, and drawn to the surface by malacates. In some of the mines, however, the ore is taken to the surface on the backs of labourers, who, with a load of from two to three hundred pounds, make several journeys daily from the bottom of the mine, often a depth of from four to five hundred varas, to the surface. At the mine of Mellado an inclined railway is employed for this purpose. When brought to the surface, the ore is conveyed to the mine yard, where it is placed in separate heaps under the care of the buscon, who picks out a further amount of sterile matter, and prepares the ore for sale. One day in each week is generally fixed for the sale of the ores, and the miners may then be seen laying out their several parcels, so as to make them appear to the best advantage, by turning each stone in such a way as to expose the best side of it, and keeping his lot constantly sprinkled with water, which has the effect of making the mineral look darker and richer.

The various purchasers, or *rescatadores* as they are called, take, as nearly as possible, an average sample from each lot, which is ground into a fine powder, and subjected to a sort of assay on the spot. This is effected by washing a handful of the pulverised ore in a *jicara* or small bowl, formed of half a gourd; and after washing away the lighter earthy particles, the richness of the parcel is estimated in accordance with the amount and appearance of the mineral deposit remaining in the bowl. Long practice enables those who constantly employ this process to arrive at results very closely approximating to the true yield of the ores, but many of the purchasers now prefer to regulate their bids by the result of fire assays, and in this case the samples are taken on the day previous to that on which the sale is to take place.

When the hour of sale arrives, a bell strikes as a signal that it is about to commence, and the person entrusted with its management takes his place successively at the foot of every parcel. Each purchaser, in turn, now approaches, and whispers in the ear of the salesman his bid for the parcel at the foot of which he stands; and when all have given in their prices, the lot is adjudged to be the property of the highest bidder. When two persons offer the same price, the first bidder obtains the parcel. The salesman and purchasers thus move from lot to lot, until the whole of the ore in the yard has been disposed of, after which it is removed by the various buyers at their own expense. The quantity of ore contained in each parcel is not ascertained by weighing, but, like the assay by washing, is determined with great accuracy, by the eye, by those who have had long practice in the business.

Zacatecas.—The district of Zacatecas is situated about fifteen leagues east of the elevated mountains which join the principal chain of the cordillera extending from Durango to Guanaxuato. Burkart, who lived many years in this vicinity, and who had consequently ample opportunities of becoming acquainted with the geology of the country, considers that the greenstone which encloses the greater number of the veins in that locality can only be regarded as a diorite. Bustamante, on the contrary, calls it syenite. This rock, which at some points presents many of the characteristics of chloritic slate, overlies a blue argillaceous slate which occasionally comes to the surface, and closely resembles that forming the enclosing rock in the deepest portions of many of the Mexican mines. Quartz veins are exceedingly numerous in the neighbourhood of Zacatecas, and have generally a direction approaching north-west and south-east. The Cantera vein is remarkable, not only on account of its variation in direction at different points in its course, but also for its prominent outcrop, which may be traced on the surface over a length of more than three leagues. This outcrop, which sometimes stands at a height of a hundred feet, and is from forty-five to sixty feet in width, throws off a branch towards the north, which is known as the San Martin vein; whilst the principal lode, turning towards the south, takes the name of the Del Muerto, and finally makes an intersection with the Quebradillas. This vein, which has a dip of 35° towards the south, is for a long distance enclosed between a greenish slate on its foot wall, and sandstone on the hanging wall, but finally enters the green slate, which thus constitutes the enclosing rock on either side. The principal workings on this vein, which is at this point about 36 feet in width, produced large quantities of ore, yielding by amalgamation from 0·0010 to 0·0012, or from 33 to 39 oz. of silver per ton; but the workings having been badly secured, fell in some years since, and caused the abandonment of the mine.

The San Bernabé vein was first opened by the conquerors of Mexico in 1548, and is said, in common with the San Alvado, to have been worked by Cortez. The eastern extension of this vein, which dips to the south, and of which the direction is nearly east and west, has produced large quantities of ore in the concessions of Malanoche, Rondanera, Loreto, and Peregrina; but the most productive mines of Zacatecas, in 1843, were those of San Cleménte and San Nicolas, of which the workings were commenced in 1836, to the east of the mines above mentioned. In the various workings of the old mines, the width of this vein was from twelve to thirty feet, but in the San Cleménte and San Nicolas it rarely exceeds six feet, although the selvages of clay on the walls, which frequently have a considerable thickness, are so impregnated with argentiferous matter as often to admit of being worked with as much advantage as the lode itself.

The most important vein of Zacatecas is the Veta Grande, which has been worked over an extent of nearly 2,500 fathoms. The first extensive operations on this lode were undertaken in 1765, by a

M. Laborde, a Frenchman. The general direction of the vein is north 60° west, with an inclination of about 35° towards the south, and like the Veta Madre of Guanaxuato, it is divided into bands, which sometimes have an united thickness of seventy-five feet, but of which the average width may be taken at about forty feet. Of this width about twelve feet are occupied by country rock, dividing the lode into three bands. The gossans, or colorados, usually extend to a depth of forty fathoms, and contain native silver and chloride of silver; the ores below the water level consist of various sulphides of silver, associated with a little blende, galena, and iron pyrites. A *bonanza*, or bunch of ore, discovered in this vein, and worked from 1828 to 1838, yielded a net profit of above $9,000,000 to the proprietors of the mines.

According to Bustamante and Burkart, the Veta Madre produced from 1790 to 1833, 3,902,252 marks, or about 2,500,000 lbs. of silver.*

Where first worked by the Bolaños Company, the vein-stuff, as extracted from the mine, was so generally productive, that only one-sixth had to be rejected as not paying the expense of reduction. Gradually, however, the vein became so much impoverished, that not quite two-thirds could be treated with advantage. During the former period the proportion of silver extracted by fusion was one-fifth of the whole production, whilst, during the latter, ores sufficiently rich for this method of reduction were very rarely met with.

The average produce obtained from the ores of the Veta Grande at the *Hacienda* de la Sauceda from 1804 to 1839, was as follows:—

1804 to 1808 . . .	0·0025 = 81	oz. per ton of 2,240 lbs.
1820 „ 1824 . . .	0·0021 = 68	„ „ „ „
1825 „ 1832 . . .	0·0019 = 62	„ „ „ „
1839 . . .	0·0017 = 55	„ „ „ „

It must, however, be remarked that the produce obtained by no means represents the actual richness of the ores treated, since from the large quantities of antimonial silver present, the patio process failed to extract nearly the whole of that metal, and this loss is estimated by Duport at nearly two-fifths of the total contents of the argentiferous matter subjected to amalgamation. The Veta Grande, however, like all the principal veins of Mexico, begins to diminish in richness at depths, from the surface, varying between 75 and 150 fathoms. Another important vein in the district is that of the Tajos de Panuco, on which open workings were commenced in 1548, and extended over a length of above three hundred and fifty fathoms. The silver from the Veta Grande contains but a very small proportion of gold; that from the San Bernabé often affords from 0·001 to 0·003 of this metal.

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* The Spanish mark equals 3,550·5 grains.

The following table, from Duport, will afford an idea of the produce of the mines of Zacatecas—

						Marks.		lbs. troy.
From	1st June,	1548,	to	16th September,	1810 .	67,317,937	=	42,161,224
„	16th September,	1810,	„	1st June,	1818 .	2,296,472	=	1,438,484
„	1st June,	1818,	„	„	1825 .	2,107,350	=	1,319,833
„	„	1825,	„	31st December,	1832 .	3,532,769	=	2,212,573

The operations of the English Company were suspended in 1838, after which time the produce of the mines continued gradually to diminish, and in 1842 some of them were let out to buscones at a *partido*, or share of one-fourth, which in 1846 had been increased to one-third.

Fresnillo.—The Cerro de Proaña is situated fourteen leagues north-west from Zacatecas, and consists of an elevation of about six hundred yards in length, which does not exceed three hundred feet in height above the level of the surrounding plains. This hill is composed of a compact argillaceous rock overlying, at a certain depth, a blue slate containing bands of white quartz. In these rocks there exists a considerable number of veins running nearly north and south, and generally dipping towards the west, but at such variable angles that the different veins often intersect each other in depth, thereby forming a species of network. The whole surface of the mountain is here covered by the débris of former workings, and large open cuttings are seen, forming more or less continuous trenches on the course of the different veins.

These mines are known to have been worked by open cuttings and otherwise, as far back as 1755, but about the year 1760 operations were commenced on a more extensive scale, and abundant returns obtained over a considerable extent of ground. Mining was subsequently carried on by some merchants of the city of Mexico, afterwards by the State of Zacatecas, and later under the auspices of the Central Government.

Real del Monte.—The mines of Real del Monte are situated about sixty miles due north from the city of Mexico, in a district containing numerous bands of porphyry of evidently different ages. To the north-east of Real del Monte the porphyries are covered by columnar basalt, and at a still greater distance, in the same direction, by metamorphic slate. The rock enclosing the various argentiferous veins of this locality is a decomposed porphyry, in which the presence of hornblende is announced by numerous greenish stains. The principal veins of Real del Monte run nearly east and west, whilst the cross-veins course almost north and south, the inclination of the former being to the south, and of the latter towards the west. The principal east and west vein of the Real del Monte is the Biscaina, usually about fifteen feet in width, but there are numerous others, particularly near the town of Pachuca, which have nearly the same direction, whilst these are crossed by the Santa Brigida, and others.

The partido system has prevailed among the mines of Real del Monte from a very early period, but differs somewhat in its arrangement from that of the other mining districts. Here the working miner or *barretero* receives a certain portion, frequently an eighth, of all the ore he raises, and in addition is paid a daily sum equal to about two shillings. From the Real del Monte Company having been the first of the large associations organised in this country for working mines in Mexico by the aid of British capital, some notice of its history and progress may not be uninteresting.

Real del Monte Company.—But little is known of the mines of Real del Monte, prior to the year 1749, except that they had yielded considerable quantities of silver from irregular and detached surface workings, and had at last become almost entirely abandoned from the influx of water, as the excavations increased in depth. At the above period, Don Pedro Terreros, a merchant of Queretaro, joined a practical miner named Bustamante, in a general *denuncio* of the district, by which they obtained possession of its two principal veins, La Biscaina and La Santa Brigida, on the condition of effecting their drainage by means of a horizontal gallery.

For this important work, a point was chosen sufficiently down the slope of the northern descent of the valley to ensure its entering the principal mines of the Biscaina vein at the depth of one hundred fathoms from the surface; but as the distance necessary to obtain this difference of level was nearly 1,500 fathoms, the work, although commenced in 1749, was not completed until 1759. Bustamante did not live to see the result of his great work, but Terreros persevered to reap the reward; for, having by this adit freed the mine from water, he extracted at comparatively small cost, up to his death in 1781, the large amount of fifteen millions of dollars, having been previously ennobled by the King of Spain, under the title of Conde de Regla.

His successor, the second Conde, continued the working of the mines, but not to equal profit with his father, who, having exhausted the upper portions of the veins, rendered dry by the adit, left his son the more difficult task of contending with the water under that level. This drainage was effected by malacates, which, raising the water in skin bags to the level of the adit, were for some time sufficiently effective, but gradually, as the mine became deeper, the difficulty and cost of drainage with such imperfect machinery also augmented. In the year 1801, the twenty-eight malacates then at work, occupying twelve hundred horses, with four hundred men, and costing $250,000 per annum, were not found sufficient to keep down the water to fifty-four fathoms below the adit, to which depth the workings on the Biscaina vein had attained in the mines of Santa Teresa and Guadalupe; and consequently, although these mines were then producing at the rate of $400,000 annually, the drainage was suspended, and their deeper workings abandoned. After this the workings were limited to a higher level; and on the hitherto unexplored extensions of the vein, and up to 1809, when the second Conde died, the mines of San Ramon with some others continued to yield about $300,000 per annum.

From this period the produce of the mines gradually decreased, and the War of Independence having commenced, their working was entirely suspended in 1819; the total produce since the death of the first Conde having been $10,000,000.

After the recognition of the independence of Mexico by Great Britain, the attention of English capitalists became directed to the mines of that country, and, at the suggestion of the late Mr. John Taylor, an association known as the Real del Monte Company was formed for restoring and draining the mines belonging to the Regla family.

In July 1824, Captain Vetch of the Royal Engineers, the first commissioner of the Real del Monte Company, arrived at the Mines, which he found in a state of utter ruin; most of the shafts had fallen in, leaving their former sites only to be detected by immense hollows overgrown with brushwood. A still more serious evil was the destruction of the great adit, which having in many parts so gone to ruin that it no longer carried off the water, it consequently rose to a great height in the mines. All the machinery in the large reduction works was gone, the population had become very scarce, and the town a collection of ruins. The chief inducement of the new Company to resume the drainage of these deep mines, being the advantages they expected to derive by the substitution of steam-power for the imperfect and costly one of horse machines raising water and ores in skin bags, a body of miners and mechanics, with steam-engines, pumps, and other machinery, arrived at Vera Cruz in May 1825, under the charge of the late Colonel Colquhoun of the Royal Artillery.

The rainy season commenced shortly after, attended by its usual scourge, the yellow fever, and soon made sad havoc among both English and Mexicans. Colquhoun, however, persevered through all difficulties, and by May 1826 the engines had arrived at the mines. In the meantime the district of Real del Monte had been carefully surveyed, the great adit cleared and restored, many of the shafts had been repaired down to adit, and buildings, workshops, and stores were rapidly rising around the mines. From this period the work of clearing and restoration progressed steadily, until by the end of 1829 the drainage had been effected to the depth of fifty-four fathoms under adit. The annual cost of pumping did not exceed $30,000, being an immense saving as compared with the large sum of $250,000 which the twenty-eight malacates formerly employed for this purpose had cost the second Conde when at the same depth he abandoned the lower workings.

It, however, soon became evident that the drainage at the distant points of San Cayetano and Dolores was not sufficiently effective for working the richest portions of the vein in the mine of Santa Teresa: a new vertical shaft was consequently sunk, and a more powerful engine erected. An engine of fifty-four inch cylinder having been erected, and assisted by the smaller ones at Dolores and San Cayetano, this portion of the Biscaina vein was worked for some time with considerable profit, and to the depth of 235 fathoms from the surface, or 120 fathoms under adit. At this point the water having so increased as again to overpower the engine, a still larger one of seventy-five inches was erected at Dolores. The reward of this expenditure was two bunches of rich ore, the one discovered on the Santa Brigida vein near Acosta, and called La Luz; the other at San Enrique, on the Biscaina vein near Dolores.

Up to the end of 1847, however, the general result of working the mines had been decidedly unfortunate to the English adventurers; for although they had profited by three rich bunches of ore at Terreros, Acosta, and Dolores, and had produced $10,481,475 worth of silver, the outlay on all the undertakings of the Company had also reached the amount of $15,381,633, leaving a loss of nearly five millions of dollars as the result of the twenty-three years they had held the mines.

The deep workings were now 120 fathoms under the great adit, or sixty-one fathoms below the point at which they had been abandoned by the second Conde, and the difficulty of drainage had so increased, both from the augmented quantity of water, and the greater height to which it was necessary to raise it, that three powerful steam-engines, which were discharging two thousand seven hundred gallons per minute, at a cost of $90,000 per annum, could barely stem the coming water of the mine. To show the difference of the cost of steam drainage as compared with the plan of the country, becomes at this point interesting. The English Company, at the commencement, had easily effected with two small engines, and at a cost of $30,000, what the Conde de Regla had been obliged to relinquish in 1801 with twenty-eight malacates, and at an annual cost of $250,000; but with the increased depth, and greater volume of water, three pumping engines expending $90,000 were barely able to maintain the drainage, while to replace them would have required at least one hundred and eighty malacates, employing seven thousand horses, with upwards of two thousand men, and an expenditure of not less than two millions of dollars per annum.

With this increased difficulty of drainage, seeing their rich bunches of ore all worked out, and a debt of five millions of dollars still outstanding, it is not surprising that the energy and perseverance of the English adventurers were at last exhausted. Towards the middle of 1848, Don Manuel Escandon and Don Nicanor Beistegui were induced by Mr. Buchan, the manager of the mines under the original Real del Monte Company, to take up the enterprise on terms which, although not returning to the English adventurers much of their lost capital, at least relieved them from all further responsibility.

After having carefully examined the position of the enterprise, this gentleman came to the conclusion that the concern had hitherto been worked, as well on too limited a scale as with too expensive an establishment; but particularly that without any effective attempt to render the poorer and more abundant ores available, or to make new discoveries on the higher and still virgin portions of the veins, every effort had been directed to the search after rich ores in depth, which, when at last discovered, did not repay the large amounts expended to find them. With reference to this subject Mr. Buchan remarks:—"Experience had convinced me that to render an extensive mining enterprise secure, it should mainly depend on the poor and abundant ore of its veins for the current cost of exploring them, so that the richer bunches, which would occur in a regular and systematic process of working, might be found without any forced effort or outlay, and thus become more profitable. It was further clear that as certain costs, such as general management, drainage, rents to owners, &c., were unavoidable and nearly the same under any scale of operations, that a larger return of the poorer ores must be obtained, in order to support them, and lastly that a perfect system of economy in every branch of so large an establishment, was most essential to its success."*

To carry out these views, he commenced by arranging the entire system of accounts on such a plan that every week's result, in each mine and reduction work, might be clearly shown, and the economy of the different departments thus fairly compared with each other. To reduce the excessive cost of drainage, he abandoned the very deep workings on the Biscaina vein, and only maintained the water to sixty-five fathoms under adit by one large steam-engine, while at the same time the extraction of poor ores was facilitated, and increased by additional shafts, winding machines, and internal rail-roads. Conjointly with these operations the eastern and yet virgin portions of the Biscaina vein were selected for the site of new trials, in high ground above water level, and another trial was commenced at Pachuca in the Rosario mine. Here the Company had already worked and incurred a considerable loss by extracting ores from the old mine, but finding that the famous vein of Xacal would intersect that of Rosario, to the eastward of all former trials, it was determined to take advantage of an adit, already considerably advanced, to examine so promising a point. After driving this adit some sixty fathoms, they had the good fortune to meet with the expected intersection of the veins, from which a considerable extraction of ore was at once commenced.

It is a peculiarity of the mineral veins of this district, that while those which run in an easterly and westerly direction produce ores readily reduced by the usual amalgamation process, practised to so great an extent in Mexico, the ores from the north and south veins have a different mineral character, and in many cases will not yield their silver by this process. Trials having, however, proved that they could be advantageously reduced by a modification of the well-known barrel system of Freiberg, and the great mass of the poor ores being of this character, it was determined to adopt barrel amalgamation on a very large scale. For this purpose, two new works were erected at San Miguel and Velasco, whilst those of Regla and Sanchez were enlarged and modified.

In the haciendas of San Miguel and Regla, which are at a distance of twelve miles from the mines, water-power is alone used to drive the machinery, blow the furnaces, grind the ores, and revolve the barrels; but at those of Sanchez and Velasco, built at the mouth of the valley descending from the mines, steam is employed as an auxiliary power; so that in 1855, besides the powerful pumping engine of seventy-five inch cylinder, which drained the mines, there were, at the reduction works, two smaller rotatory engines and eighteen water-wheels of different sizes, which drove 110 wet stampers, sixteen large arrastres, and three edge mills, all employed in grinding ores; besides revolving sixty-four large barrels in which the prepared ores were amalgamated. In the hacienda of Regla were eight blast furnaces for smelting rich ores; and in San Miguel, Velasco, and Sanchez, thirty-six reverberatory furnaces for drying and roasting the ores preparatory to amalgamation in barrels. We are informed by Mr. Buchan that a great increase in the reduction works has been made since 1855, by the erection of the hacienda of Loreto, which, situated in the midst of the Pachuca mines, has already the power of reducing yearly 55,000 *cargas* of ore by the patio process, and is being gradually increased, so as to be enabled to treat 100,000 cargas annually.*

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* Report of the Real del Monte Company, 1855.

The machinery employed by the Real del Monte Company having been much increased since 1855, the following are now in operation:—

Steam Engines for Drainage.							
1	of 75	in.	cylinder	on the	Dolores	Shaft.	
1	of 54	„	„	„	Acosta	„	Real del Monte.
1	of 30	„	„	„	San Patricio	„	„
1	of 85	„	„	„	San Juan	„	Pachuca.
1	of 30	„	„	„	San Nicolas	„	„
1	of 30	„	„	„	Corteza	„	„
1	of 18	„	„	„	Rosario	„	„

Total drainage — 7 steam-engines.

Rotary Engines.				
1	San Juan	Shaft,	Pachuca;	winding.
1	Rosario	„	„	„
1	San Pedro	„	„	„
1	Velasco hacienda; driving barrels.			
1	Sanchez	„		„
3	In workshops, turning lathes, blowing, &c.			

Total rotary — 8

Grand Total —15 steam-engines.

Water-wheels.				
3	Loreto	reduction	works;	one pumping and two working stamps.
1	Guerrero	„	„	wet stamps.
2	Aviadero	„	„	„
1	Sanchez	„	„	driving barrels.
1	Velasco	„	„	„
2	Peñafie	„	„	driving wet stamps.
6	San Miguel	„	„	two wet stamping, four driving barrels.
11	Regla	„	„	one „ ten working arrastres, &c.
2	Regla smelting works; blowing and stamping.			

Total 29 water-wheels.

The present grinding machinery amounts to 350 stamp heads, 24 water-power, and 50 mule-power arrastres, and 80 amalgamating barrels; which reduce yearly—

	Cargas.
By the barrel process	206,000
„ patio „	106,000
Total	312,000

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* The carga weighs 300 lbs.

The following summary of the operations of this Company for 1860, which was one of the most favourable years, the produce being good, and the amount of dead-work small, is of great practical interest:—

General Results of Mining Operations by the Real del Monte Company, in the districts of Real del Monte and Pachuca, in the year 1860.

General expenses of management	$50,170
Cost of draining the two districts	167,934
Cost of extracting ores from various mines	647,338
Cost of reducing ores in different haciendas	841,606
Duties on silver, paid to the Mexican Government	173,587
Freight of ores from mines to reduction works, including cost of repairing roads	186,503
Convoys of silver to the coast, or mint of Mexico	14,877
Agencies, Commissions, &c.	7,109
Total cost on current working of the mines	2,089,124
Total produce, 277,396 cargas of ore, from which were extracted 423,394 marks of silver, value	3,710,891
Profit on current working	$1,621,767

The above profit was applied as follows:—

Re-invested in discovery works in different mines	$181,052
Re-invested in enlarging and improving works for the reduction of ores	73,120
Purchase of forests for fuel	31,000
Cost of maintaining a force of 150 cavalry and 50 infantry, necessary for the security of the districts during the civil war	60,000
Paid as dues to part owners of mines	353,070
Paid as dividends to the Shareholders of the Company	923,525
Total	$1,621,767

Table of stores consumed during the year 1860:—

From forests belonging to the Company—		
Timber	$ 30,000	
Wood-fuel	200,000	
Charcoal	60,000	
		$290,000
Salt	$150,000	
Quicksilver	100,000	
Iron and Steel	50,000	
		300,000
Carried forward		$590,000

Brought forward		$ 590,000
Barley and Straw	$ 100,000	
Tallow and Oil	40,000	
Gunpowder	15,000	
Sulphate of Copper	14,000	
Sacks and Cordage	18,000	
Lime and Bricks	10,000	
Litharge	13,000	
Leather and Hides	15,000	
Sundry stores	45,000	
		270,000
Total		$ 860,000

Table *showing the details of Silver Ores reduced by the Real del Monte Company, in the districts of Real del Monte and Pachuca, from May* 1849, *when the present Company was first formed, to the end of the year* 1865.

	Ores reduced by smelting.		Ores reduced by Barrel amalgamation.		Ores reduced by Patio amalgamation.		Total each Year.			
To Dec.	Ores.	Silver.	Ores.	Silver.	Ores.	Silver.	Total Ores reduced.	Total Silver produced.	Total Value of Silver produced.	Mean Produce of Silver per monton.*
	cargas	marks.	cargas.	marks.	cargas.	marks.	cargas.	marks.	dollars.	marks.
1852	7,108	45,671	281,629	221,827	21,975	19,412	310,712	286,910	2,508,675	9·00
1853	2,903	15,358	141,208	131,518	37,040	29,063	181,151	175,939	1,537,796	9·71
1854	2,386	14,913	152,614	160,900	37,982	31,313	192,962	207,126	1,811,822	10·73
1855	2,690	23,612	181,353	217,193	38,010	46,868	222,053	287,673	2,375,503	12·90
1856	6,011	48,666	209,053	243,041	46,490	58,843	261,554	350,550	3,081,663	13·40
1857	4,926	44,942	219,326	238,041	50,400	63,183	274,652	346,166	3,039,019	12·50
1858	5,056	49,582	217,461	218,291	48,355	53,638	270,872	321,511	2,824,831	11·80
1859	4,813	52,057	226,775	283,112	44,013	51,434	275,601	386,603	3,404,459	14·00
1860	4,698	47,442	222,498	314,745	50,200	61,207	277,396	423,394	3,710,891	15·20
1861	4,162	42,397	212,480	263,990	63,947	87,545	280,589	393,932	3,782,399	15·70
1862	4,203	47,518	209,861	310,906	60,412	73,968	274,476	432,392	3,445,222	14·00
1863	3,543	32,983	194,097	217,204	76,920	88,370	274,560	338,557	2,984,351	12·30
1864	3,500	29,067	205,850	229,946	105,740	142,697	315,090	401,710	3,432,107	12·40
1865	1,921	19,273	170,606	186,535	97,642	144,055	270,163	349,863	3,044,572	13·00
	57,920	513,481	2,844,805	3,237,249	779,126	951,596	3,681,851	4,702,326	40,983,290	12·77

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* The *monton* varies in the different mining districts, but, in Real del Monte and Pachuca, is 10 cargas, or 3,000 lbs. The mark of silver is 8 Spanish oz. of 443·8 gr. each.

Produce *and* **Profit** *of the Silver Mines worked by the Real del Monte Company, in the districts of Real del Monte and Pachuca, Mexico.*

	Value of Silver produced.	Profits paid.		Duties on Silver to Mexican Government.	Cost of draining Mines.
		As Dues to Part Owners.	In Dividends to Shareholders.		
Four Years, to	dollars.	dollars.	dollars.	dollars.	dollars.
Dec. 1852	2,508,655	155,373	......	101,109	215,541
1853	1,537,796	152,681	256,250	66,015	69,344
1854	1,811,822	199,371	307,500	82,566	83,707
1855	2,375,503	194,511	461,250	108,604	86,815
1856	3,081,663	276,652	820,000	140,441	115,886
1857	3,039,019	241,553	461,250	138,375	107,286
1858	2,824,831	233,294	410,000	130,860	128,177
1859	3,404,459	364,858	666,250	158,939	120,777
1860	3,710,891	448,905	871,250	173,587	167,934
1861	3,782,399	433,963	820,000	175,359	156,627
1862	3,445,222	341,018	334,500	159,620	143,075
1863	2,984,351	178,936	557,500	138,466	174,135
1864	3,432,107	279,377	580,000	159,437	178,856
1865	3,044,572	136,815	......	141,064	212,713
	$40,983,290	$3,637,307	$6,545,750	$1,874,442	$1,960,873

Total dividends to Shareholders of Real del Monte Company . .	6,545,750
Paid as dues to part owners of Mines	3,637,307
Duties to Mexican Government	1,874,442*
Total profit of the Real del Monte and Pachuca mines, 17 years	$12,057,499

Mr. Buchan, to whom we are indebted for the foregoing statistical information, remarks :—

"That no dividends were paid to shareholders of the Company in 1865, must not be attributed to any failure of the mines themselves ; but that during the very unsettled state of Mexico, our expenses were greatly augmented by the increased price of materials, and the loss incurred in keeping up a considerable military force for the security of the district. Contributions and forced loans to different governments have for several years past been exceedingly heavy ; but with the hope of their speedy repayment, these sums have been carried to a 'Suspense Account,' until the amount became so large as to determine the Company to suspend dividends, and dedicate all their share of profits to the liquidation of this debt.

"All the accounts I receive from Mexico assure me that the state of the mining negotiation is as promising now, as at any former period.

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* Legal dues, not including forced loans.

"The great formation of ore in the Rosario vein of Pachuca, which, since its discovery in 1850, has yielded 25 millions of dollars, still promises a steady continuance of its produce; and although by underlie south, and extension south-eastward, this orey ground is now passing gradually out of the old mine of Rosario into the neighbouring setts of San Pedro and Guatimotzin, to the benefit of the owners of these latter mines, yet this will in no way affect the Company, whose interest is equal in all.

"The mine of Xacal, on the same vein, and adjoining Rosario to the west, which in former times was so famous for its richness, and the chief inducement for the drainage of the Pachuca district, has hitherto proved a great disappointment, by being found poor in its lower workings. But the works of discovery which have been steadily prosecuted into new ground, are now opening out what appears so rich and extensive a body of ore, as promises to entirely change the character of this mine. Many of our other mines are also in good produce; and if the unsettled state of Mexico does not interfere with our mining operations more than it has hitherto done, the coming year promises to be a very profitable one in Real del Monte, but more particularly so in the Pachuca district."

NEVADA.—Although the discovery of silver in this region can only be said to date from 1859, its extraordinary production has already rendered it more famous for its mineral wealth than localities in which this metal has been for centuries continuously mined in large quantities. This abundance of mineral wealth has attracted a numerous and industrious population to a country before but thinly inhabited by tribes of wandering savages; and flourishing towns have already sprung up in a desert which, from its situation and inhospitable character, seemed for ever to defy the progress of civilization and the arts.

The two principal mining centres of Nevada are Virginia city in Storey County, and Austin in Lander County; and, although numerous veins are being more or less extensively worked in other portions of the State, the mines in the vicinity of these towns have hitherto been more productive than those of any other mineral region.

Great Comstock Vein.—The range of the Washoe mountains, in which the Great Comstock Vein is situated, is separated from the eastern slope of the Sierra Nevada by a continuous meridional depression, in which are the deep basins of the Truckee, Washoe, and Carson valleys. Its shape is irregular, but its general direction is from north to south. In a southerly direction, it gradually slopes down to a smooth table-land, traversed by the Carson river, beyond which the Washoe mountains become merged into the more elevated Pine-nut range. Towards the west, the Washoe hills descend rapidly, and finally sink beneath the detrital beds of the Washoe and Truckee valleys, but are connected with the Sierra Nevada by two low granitic ridges, stretching across the northern and southern extremities of Washoe valley. To the north-east, this range passes into a very extensive, and but little explored, mountain region; whilst to the south-east it abruptly disappears below one of the basins of the Carson river. The entire width of this range is not above fourteen miles, whilst its length is not yet determinable, on account of the scanty knowledge possessed of the northern portions of the State. The culminating point of the Washoe range is Mount Davidson, the elevation of which, as determined by Whitney, is 7,827 feet; and at its foot are situated Virginia city, and the principal mines on the Comstock lode.

The aspect of the Washoe mountains is exceedingly barren, as is also the view from Virginia, over the hilly country to the east. The air is, however, extraordinarily pure and transparent, so that every ravine and declivity on the sides of mountains a hundred miles distant is readily distinguished. At the time of the discovery of the Comstock vein, the Washoe mountains were covered by scattered and stunted trees, of nut-pine and cedar; but these have long since disappeared, and Virginia now depends for her supply of wood on the slopes of the Sierra Nevada.

The enormous consumption of wood for fuel, and of square timber in the mines, is, however, rapidly causing the destruction of these forests; and the time is, consequently, not distant when the mines and reduction works of Virginia must derive their wood from greater distances, and chiefly from near the Truckee River, which is about thirty miles from the mines, and to which a railway, in connexion with the Central Pacific Railroad, is projected. Mount Davidson, the prominent central point of the Washoe range, consists of syenite, which is here composed of two kinds of feldspar, orthoclase and oligoclase, associated with hornblende, mica, and occasionally epidote. Metamorphic rocks adjoin the syenite on the north and south, and are intersected by dykes of that rock, which of itself sufficiently proves its later origin. These metamorphic rocks differ materially in their lithological characteristics, but may be divided into three distinct groups, the most recent of which belongs to the triassic epoch. These are immediately preceded in age by a series of micaceous and quartzose slates, frequently containing bands of limestone; below which is a third series, chiefly consisting of hornblendic rock, with interstratified layers of quartzite, grey slate, crystalline limestone, and specular iron. These rocks form the hills which flank American Flat to the west, as well as those between Silver City and Carson, and are generally capped by an overflow of porphyry. They constitute the *ancient series*, and partially preceded, and were partly contemporaneous with, the gradual upheaval of the Sierra Nevada and the entire chain of the Cordilleras. The *recent series*, which is volcanic and eruptive, belongs to the latter part of the tertiary and beginning of the post-tertiary periods.

The first of these in point of age is a species of dioritic porphyry, to which Richthofen has applied the name of *propylite*. This rock has the peculiarity of exactly resembling many ancient rocks in appearance, whilst in reality it is of very recent origin. It occupies a prominent position among the enclosing rocks of the Comstock vein, and likewise encloses a considerable proportion of the largest and most productive silver veins in various parts of the world. Among these may be mentioned the silver veins of the Carpathian Mountains, those of Zacatecas, and other localities in Mexico, and probably, also, many of those in Bolivia. This rock is composed of a fine-grained paste, generally of a greenish, but sometimes of a red, grey, or brown colour, with embedded crystals of oligoclase, and columns of dark-green fibrous hornblende, which is also the colouring matter of the base itself. A peculiarity of this rock is its ferruginous character, when decomposed by weathering. Geologically, it is an eruptive rock, but has been accompanied by vast accumulations of brecciated matter, which are sometimes found regularly stratified. Several different kinds of volcanic and eruptive rocks followed the outbreak of propylite; but of these, trachyte is the most important, not only as taking a prominent part in the formation of a large extent of country, but also on account of its intimate relations with the formation of the Comstock vein. Its essential characteristic is the predominance of glassy feldspar, which, along with hornblende and mica, is embedded in a paste of a peculiarly rough texture, caused by the presence of an infinity of microscopic vesicles.

Eruptions of basaltic rock, of considerable magnitude, have taken place in various parts of the great basin; but in the neighbourhood of the Comstock vein they have not made their appearance to any important extent. Volcanic and eruptive agency appears to have gradually declined in the whole region, leaving the last trace of its active existence in the still boiling waters, and daily increasing silicious deposits, of Steamboat Springs.

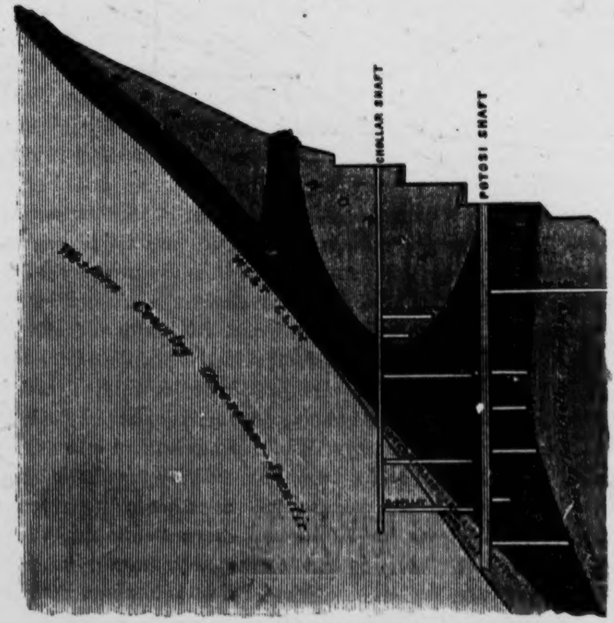
The Comstock vein runs nearly in the direction of the magnetic meridian, along the eastern slope of the Mount Davidson range, a little above the slope on which are built the towns of Virginia, Gold Hill, and American City. The outcrops extend in a broad belt along the foot of the mountain, and immediately at the back of the three towns. The course of the vein, as far as it has been hitherto explored, appears to be more or less modified by local circumstances, passing the ravines in concave bends, and enclosing the foot of the different ridges in convex sweeps. These irregularities are of considerable importance, as they would seem to influence the ore-bearing character of the lode. The length to which the Comstock vein has been traced with certainty is somewhat over nineteen thousand feet, while its total extent is probably at least twenty-four thousand feet. The most extensive operations have been carried on between the Ophir Mine, north, and the Overman, a distance of about eleven thousand feet; whilst in other parts the workings have been of a very superficial character. Several of the mines are worked to a depth of six hundred feet; but the shafts of the Hale and Norcross, Chollar Potosi, and Gould and Curry, have reached a depth of above eight hundred feet.

The Comstock vein, at a depth of from four to six hundred feet below its outcrop, has a width of from one to two hundred feet; but contracts in some places, so that its two walls come into close proximity with each other. At this depth, both of its walls descend, with an easterly underlie, at an angle varying from forty to sixty degrees. Upwards, from an average depth of five hundred feet, the western wall rises to the surface with the same inclination; whilst the eastern assumes a vertical position, and subsequently turns off with a westerly dip. The vein consequently expands towards the surface in the shape of a funnel, as shown in Fig. 39, which represents, on a scale of 240 feet to one inch, a section made by Messrs. James and Stretch, Mining Engineers of Virginia, through the Chollar and Potosi shafts.

This increase in volume is especially produced by the intervention, between the vein-matter, of large masses of country rock, detached from the walls, but which have usually moved a short distance only downward, by sliding from their original positions. The number and bulk of these *horses* increase towards the surface, where some of them have a length of nearly a thousand feet, and in many cases a width of above a hundred. The vein-matter filling up the spaces between these intervening masses of country rock is generally, near the surface, far inferior in width to that of the horses lying between it. The width of the mineral belt in which these branches come to the surface, and there form a scattered outcrop, is usually above five hundred feet.

The Comstock lode is, on the western side, accompanied by a number of smaller veins, the outcroppings being visible on the surface, and some of which appear to be of considerable size. The western boundary of the main vein is defined by a continuous clay selvage, or flucan, lying on the smooth foot wall, and separating the vein-matter from the country. On the eastern sides, the adjoining country rock is impregnated with matter similar to that composing the vein itself; and consequently, the well-defined east wall has often not precisely the same relative position with regard to the entire vein, and, by

FIG. 39.



SECTION OF COMSTOCK VEIN.
(*Looking North.*)

cutting through what appears to be the east wall, another or succession of others of a similar character are discovered beyond it.

The enclosing rocks which accompany this vein vary somewhat in different portions of its course, although on the eastern side they invariably consist of slightly-differing varieties of propylite, or compact porphyry. On the western wall, the composition of the country presents greater differences. From the Best and Belcher Mine to Gold Hill, it is formed of syenite, which is, in some places, separated from the selvage of the vein by a fine-grained crystalline rock, of a black colour, partaking of the nature of aphanite, but of which the mode of occurrence is at present obscure. The outcrops of the Comstock vein do not form one continuous line, but rather consist of detached parallel ranges of brownish quartz, ordinarily protruding above the surface of the ground, and sometimes forming bold crests, which in the aggregate constitute a broad, interrupted belt. The porous nature of this quartz, originating from the decomposition and removal of fine particles of disseminated ore, and its brown colour, caused by the presence of metallic oxides, indicate the ore-bearing character of the vein in depth. A minute examination also reveals the occurrence of finely-disseminated metallic gold and silver, and occasional spots of the chloride and sulphides of the latter metal.

The vein-matter of the Comstock lode is exceedingly varied in its character; its chief constituents being fragments of the enclosing rock, clay and clayey matter, quartz, sulphide of silver, and other minerals. The shape and extent of the fragments of country rock are very various. They are, however, generally of a lenticular form, and differ in their dimensions from large "horses" down to the smallest brecciated fragments. Clay forms, in continuous sheets from north to south, the eastern and western selvages, which are sometimes from ten to twenty feet in thickness. Other sheets of this substance divide the horses from quartz, or the different bodies of quartz from each other; and where the vein-matter is so thin that the two clay bands come nearly together, the selvages have often a united width of from fifty to sixty feet.

The quartz of the Comstock lode is very rarely solid, and consequently its removal seldom requires the intervention of gunpowder; it is usually much fractured, and in many places the effects of mechanical or chemical action are so great as to give it the appearance of crushed sugar. It almost always occurs in this condition when enclosed between two clay walls, and in the immediate vicinity of rich masses of ore. The principal silver ores are stephanite, vitreous silver, native silver, ruby silver, horn silver, and polybasite, with which are associated small quantities of exceedingly rich argentiferous galena. Besides these are found native gold, iron and copper pyrites, blende, and carbonate and phosphate of lead; the two latter only occurring in very minute quantities. These ores are seldom crystallised; and consequently, specimens which would in other mining districts attract no attention, are regarded in Virginia as great curiosities. Quartz is, in reality, the only gangue of the Comstock vein. Carbonate of lime but seldom accompanies it, whilst the other carbonates are of exceedingly rare occurrence. Sulphates abound in the waters issuing from the mines, but the only one forming an essential constituent of the vein is gypsum, which is sometimes found in large crystals.* Sulphate of baryta has never been observed, and small specimens of stilbite and chabasite are only met with in the northern portion of the vein.

The ore is differently distributed in the northern and southern portion of the lode, the passage between the two methods of occurrence being very gradual. In the northern part, the ore is concentrated in elongated lenticular masses, of which the greater axis is nearly ver-

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* An examination of the water issuing from the Comstock vein, in the Ophir Mine, January 12th, 1865, afforded Mr. G. Attwood the following results :—

Heat of water, 70° Fahr., the air near being 62° Fahr.; in Seventh Gallery, 25 feet below where water was taken, and 400 feet below the surface, it was 48° Fahr. Flow of water estimated at from 150 to 200 gallons per minute; the total amount of water taken from mine being 300 gallons per minute.

One imperial gallon contained 14·84 grains of mechanically suspended matter.

Both water and suspended matter were apparently free from any trace of the precious metals.

Specific gravity at 60° Fahr., 1·0007.

Carbonic Acid in an imperial gallon, 23·30 grains.

Fixed Matter „ „ „ 28·02 „

	Composition of Fixed Matter.
Carbonate of Lime	4·10
„ Magnesia	2·82
Sulphate of Lime	10·01
„ Potassa	0·65
„ Magnesia	2·93
„ Soda	2·67
Carbonate of Iron	1·82
Chloride of Sodium	0·60
Alumina	Trace
Silicic Acid	2·20
	27·80

tical, but dips towards the south, and sometimes also to the east, their width varying from fifteen to fifty feet. In some cases, several of these adjoin each other in such a way that the most westerly one extends further north than the one situated next to it towards the east, and this again further than its next eastern neighbour. The great value of the mines on the southern portion of the vein consists in the continuous character of the ore-bearing ground, which extends for an uninterrupted distance of above fifteen hundred feet. The vein is by no means productive throughout its entire width, but the ore is concentrated in continuous sheets, the principal of which is very near to, and parallel with, the eastern wall. Its widest places have a thickness of about fifty feet; it generally commences at a depth of from one hundred and fifty to two hundred and fifty feet from the surface, and is at many points still worked in the lowest levels of the mines. Besides this very extensive eastern body of ore, there is another in the Gold Hill Mines, further to the west, which extends from the outcrop down to a depth of two hundred and fifty feet. A similar body has been worked in the Yellow Jacket, Crown Point, and Belcher Mines, and was remarkable from its southern portion being very rich in gold. It is, however, a well known fact, that the richness of the ores from the Comstock vein has in general diminished, although the total yield of the district has been more than kept up by the larger amounts which are now daily passed through the mills. When the vein was first opened, ores affording from one to seven hundred dollars to the ton were of frequent occurrence, and considerable shipments were made of parcels yielding from two to three thousand dollars per ton. Such ores are now but rarely met with, and the general average yield of all those treated will probably not much exceed thirty dollars to the ton. The cost of extraction has, however, been greatly reduced, and the methods of reduction much improved; and therefore ores which formerly could not have been treated with advantage now afford highly remunerative returns. The aggregate amount of ore now daily extracted from the Comstock vein is about twelve hundred and fifty tons.

During the early period of the workings on the Comstock vein, the relative proportions of gold and silver yielded by the ores were found to change as a greater depth was obtained, the yield of gold being greatest near the outcrops, and gradually diminishing in the deeper portions of the lode. Recently, however, the proportion of gold has again been on the increase, and this is found to be the case, not in any one particular mine only, but throughout the whole extent of the lode.[*]

The various phenomena connected with the occurrence of silver ore in the Comstock vein have been summarised by Richthofen as follows:[†]—

1st. The ore is, in the northern part of the vein, concentrated in chimneys dipping to the south; in the southern part it forms continuous sheets of great length, but which are comparatively narrow.

2nd. These deposits of ore are enclosed in the eastern, and sometimes also in the middle portions of the vein; the western branches are either barren or poor.

3rd. The richest and largest deposits have been found at those places where the outcrops, including those of the western branches, were most prominent.

4th. In the northern part the vein is, at the levels explored, invariably poor where it passes a ravine; but in the southern part the ore continues in the ravines.

5th. The richest portions of the lode are south of each ravine crossed by the vein.

6th. All the chimneys in the northern part are at those places where the walls, after close contact, rapidly diverge, and cause the vein to expand.

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[*] The following assays from parcels of crushed ores worked in barrels, by Mr. G. Attwood, show the relative proportions of gold and silver in the rock from the Comstock Mines:—

Second Class ore from the Ophir Mine, June 16th, 1863.

Silver, per ton	71·92 oz.	$93·49
Gold "	2·79 "	57·66
Value		$151·15

Second Class ore from the Gould and Curry Mine, May 12th, 1864.

Silver, per ton	81·16 oz.	$105·50
Gold "	2·58 "	52·70
Value		$158·20

Second Class ore from the Savage Mine, May 12th, 1865.

Silver, per ton	76·18 oz.	$99·03
Gold "	1·57 "	32·45
Value		$131·48

[†] "The Comstock Lode, its Character, and the probable Mode of its Continuance in Depth," by Ferdinand, Baron Richthofen. San Francisco, 1866.

7th. All the principal accumulations of ore are at those places where there was most room in the fissure for the deposition of quartz, and they are therefore generally rare where an unusual number of horses obstruct the vein.

The first discovery of silver in Nevada appears to have been made in 1857, by two brothers of the name of Grosh, in a quartz vein now held by the Kossuth Gold and Silver Mining Company, on which they had a claim. Shortly after the discovery, one of the brothers accidentally wounded himself with a pick, from the effects of which he died; and the other brother went to California, where he died early in 1858, which probably prevented the valuable nature of their discovery from becoming generally known. In the meantime, placer mining was carried on to a considerable extent in various localities, principally in Gold Cañon. In 1857, a man named Kirby, and others, commenced placer mining in Six Mile Cañon, about half a mile below where the Ophir works now stand, and worked at intervals with indifferent success until 1859. On the 22d of February, 1858, the first quartz claim was located in the Virginia Mining District, on the "Virginia Croppings," by James Finney, generally known as "Old Virginia," from whom the city of Virginia takes its name. This may be considered the first location of the Comstock lode. The discovery of rich deposits of silver ore was not made until June 1859, when Peter O'Reilly and Patrick McLaughlin, while engaged in gold washing, on what is now the ground of the Ophir Mining Company, and near the south line of the Mexican claim, uncovered a rich vein of sulphide of silver in an excavation made for the purpose of collecting water to use in their rockers in washing for gold. This discovery being on ground claimed at the time by Kirby and others, Comstock was employed to purchase it, and Comstock's name has thus been given to this great lode.

From this discovery has resulted the almost miraculous growth of the district. Claims were at once taken up on the vein for miles in extent; an unparalleled excitement followed, and miners and speculators rapidly congregated in the hope of obtaining a share of the reported wealth, while prospectors were busily examining almost every part of the country in search of silver ores.

The following table, from the report of the State Mineralogist for 1866, gives the names of the various mining claims on the Comstock lode, as far as its continuity has been ascertained; also, the length of the individual claims, length of each claim explored, percentage of explored and unexplored ground, depth of the lowest workings, and other information.*

NAME.	Length of Claim in feet.	Length explored in feet.	Per cent. of Claim explored.	Per cent. unexplored.	Depth of lowest Workings.	REMARKS.
					feet.	
Utah	1,000	300	30	70	200	Engine removed; not working.
Allen	925	300	32	68	200	Not working.
Sierra Nevada	1,959	400	25	75	650	
Union	500	5	1	99	80	Explored by tunnel; not working.
Ophir North Mine	1,200	400	33⅓	66⅔	549	Explored through Ophir, "Mexican Shaft."
Mexican	100	100	100	...		
Ophir South Mine	200	200	100	...		
Central	150	150	100	...	620	
California	300	300	100	...	428	Explored by tunnel and winzes.
Central, No. 2	100	100	100	...	369	Explored by whim on White & Murphy's claim, and by the Latrobe tunnel; not working.
Kinney	50	5	10	90	369	
White and Murphy	210	210	100	...	369	
Sides	500	200	40	60	500	Not working.
Best and Belcher	250	250	100	...	469	Engine removed; not working.
Gould and Curry	1,200	921	100	...	900	
Savage, Old Shaft	771	771	100	...	614	
,, Curtis ,,	...	...	...	...	448	
Hale and Norcross	400	400	100	...	783	
Chollar Potosi	1,434	700	50	50	923	
Bullion	940	450	47	53	803	
Exchequer	400	...	...	100	540	
Alpha	278¼	278¼	100	...	680	
Apple and Bates	31¾	31¾	100	...	Average 650 feet.	
Imperial Alta	118	118	100	...		
Bacon	45	45	100	...		
Empire North Mine	55	55	100	...		
Eclipse	30	30	100	...		
French	20	20	100	...		
Empire South Mine	20	20	100	...		
Plato	10	10	100	...		
Bowers	20	20	100	...		
Piute	20	20	100	...		
Winters and Küstel	30	30	100	...		
Consolidated	21	21	100	...		
Rice Ground	13¼	13¼	100	...		
Imperial, H. and L.	65¾	65¾	100	...		
Challenge	50	50	100	...		
Confidence	130	130	100	...		
Burke and Hamilton	40	40	100	...		
Yellow Jacket	943	943	100	51	560	
Kentuck	93¾	93¾	100	...	400	
Crown Point	540	540	100	...	400	
Belcher	940	940	100	...	850	
Segregated Belcher	160	160	100	...	500	
Overman	1,200	700	60	40	711	
North American	2,000	—	—	—	300	Not working.
Baltimore American	2,000	500	25	75	300	Not working.

North of the Utah, locations have been made on what is supposed to be the Comstock lode, but the developments are unimportant; the same remark applies to that portion of the vein south of the Overman. As far as is at present known, these two claims limit the productive portions of the vein, and include a length of nearly four miles.

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* This table commences with the claim on the extreme north of the vein.

The length given as belonging to the Best and Belcher, and Gould and Curry Companies, is the amount of ground claimed by each. The actual length of ground between the Sides and the Savage Companies, is only one thousand one hundred and forty-three feet. The lengths given for the Hale and Norcross, and Savage Companies, are measured on the croppings. Owing to the divergence of the north line of the Savage and south line of the Hale and Norcross, to the eastward, and the direction in which the vein dips, the length of their claims on the lode, in depth, is constantly increasing. Six feet of ground is in dispute between the Apple and Bates, and Imperial Companies.

There are consumed annually by the Virginia Companies about 22,650 cords of wood, at an expense of $16 per cord, and a total cost of more than one-third of a million of dollars, and they also use about 15,504,120 feet (board measure) of timber and scantling, all of which must be transported long distances in wagons, at a cost of about $40 per thousand feet. Thus, for wood and timber alone the total annual expenditure is at least one million of dollars.

Work commenced in earnest upon the Comstock lode about five years since, and its total yield has been about $65,000,000.

The following was the production of the Gould and Curry Mine from the incorporation of the Company in June 1860, to Nov. 30th, 1866:—

DATES.	Ore worked, &c.	Bullion, &c.	Average Yield.
	Tons.		Per Ton.
From July 1, 1860 to Dec. 13, 1860	140½	$22,004·82	$156·62
" Dec. 14, " " Dec. 13, 1861	300	44,221·44	147·40
" Dec. 14, 1861 " Nov. 30, 1862	8,442½	842,538·80	99·80
" Dec. 1, 1862 " Nov. 30, 1863	48,745	3,902,912·64	80·07
" Dec. 1, 1863 " Nov. 30, 1864	66,477¾	4,798,124·90	72·18
" Dec. 1, 1864 " Nov. 30, 1865	46,022¾	2,026,172·57	44·02
" Dec. 1, 1865 " Nov. 30, 1866	60,417½	1,690,952·25	28·00
Total	...	13,326,927·42	
From Tailings	...	300,143·76	
Worked, Tons	230,546	$13,627,071·18	$59·02

On hand, December 1st, 1866 . . . 4,249½ tons.
Total produce . . . . 234,795½ "

It has been estimated that five or six other Companies have taken out more than a million dollars each, and about twenty have each taken out small sums.

Richthofen estimates the total yield of the Comstock vein, up to the end of 1865, as follows; thus making the produce of silver from this source during three years, equal to about 23 per cent. of the whole production of the world during that period :—

The quantity of ore hoisted during the quarter ending 30th September, 1865, was 71,000 tons, or about 284,000 tons per annum, but this was considerably exceeded in 1866.*

Year.		Total.		Silver.		Gold.
1862	about	$ 4,000,000	about	$ 2,500,000	about	$1,500,000
1863	„	12,000,000	„	8,000,000	„	4,000,000
1864	„	16,000,000	„	11,000,000	„	5,000,000
1865	„	16,000,000	„	11,250,000	„	4,750,000
Total produce 1862 to 1865		$48,000,000		$32,750,000		$15,250,000

The aggregate amount of water daily pumped from the several mines cannot be ascertained, since no record has, in most instances, been kept. Nearly every mine has its pump, which is in most instances idle during a large portion of the twenty-four hours, but in the deeper mines it is kept constantly working. The Best and Belcher, and Hale and Norcross, each, during some part of the summer of 1865, pumped 15,000 gallons of water per hour. The Gould and Curry has a pumping capacity of 25,000 gallons per hour.†

The chief difficulties attending the working of the Comstock vein are occasioned by its immense width, and the friable and clayey nature of its constituents. In order, therefore, to be enabled to remove the ore, and at the same time keep open the necessary workings, the whole width of the lode has to be timbered with rectangular beams of pine, generally 10 in. × 12 in., which are put together in bays, about eight feet from centre to centre, in which the vertical supports stand perpendicularly over each other, from bottom to top, whilst the cap-pieces of one set, which form the foot-pieces of the next, extend across the vein from one wall to the other. The spaces between the timbers are filled in with attle when no longer required to be kept open for mining purposes or ventilation. The expansive force of the clay selvages is, however, under the influence of moisture, exceedingly great; and it is by no means uncommon, in wet situations, to see heavy beams, 12 in.×14 in., broken like match-wood, or bent so as to obstruct the levels. The subdivision of the Comstock vein into a large number of claims of very limited extent, has been productive of much unavoidable expense; occasioned not only by the multiplicity of shafts and the extra amount of machinery thereby rendered necessary, but also further augmented by the additional expense of greatly-divided management, and a series of ruinous and vexatious lawsuits, due to disputed claims and alleged encroachment of rights.

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* Tons of 2,000 lbs. each.

† The American gallon is equal to 0·83311 imperial gallon.

In order to relieve the various mines on the Comstock lode from the constantly increasing expenses of drainage and of hoisting the ores to the surface, it has been recently proposed to bring in a deep adit, sufficiently wide for a double line of railway, from the Webber Cañon, a distance of nearly four miles. This proposed gallery, which is known as the "Sutro Tunnel," would intersect the lode at a depth of one thousand nine hundred feet below its outcrop; and besides effecting the drainage of all the mines to that level, would cross-cut several veins in its course, and afford means of transport for the ores to Carson River, where water-power can be obtained, and wood procured at a comparatively cheap rate. Intervening cañons, about three-quarters of a mile apart, afford facilities for sinking four different shafts to the level of the proposed tunnel, and from these the work would be extended in both directions, as well as from its mouth in the Carson valley. These shafts would respectively require to be sunk to the following depths, viz., 443, 980, 1,360, and 1,436 feet; the cost of the tunnel is estimated at $1,983,616. It is calculated that under favourable circumstances this great level could be completed within three and a half years after its commencement; but, although the rock through which it would have to be driven is, generally speaking, of a not unfavourable character for the prosecution of such work, it is more than probable that a considerably extended time would be required for its execution. A franchise has also been granted to a company to construct a railway from Virginia to the Truckee River, with a branch from thence to Carson city. The length of the road from Virginia to the Truckee will be forty-five miles, and its maximum gradient seventy-two feet per mile. This road, complete and ready for traffic, including rolling stock, is estimated to cost $3,774,000, whilst it is stated that its gross income would exceed $2,000,000.

With the exception of those on the Comstock vein, some of the most important mines of Nevada are comprehended within what is known as the Reese River Mining Region, for much important information relative to which, we are indebted to voluminous notes furnished us by Mr. A. Blatchly, Mining Engineer, of Austin, who has devoted much time and attention to the study of the mineral deposits of that portion of the State in which he resides.

Reese River Mining Region.—The extensive mineral country known as the Reese River Mining Region, is situated in a great mineral belt which extends three hundred miles north and south, by two hundred and fifty east and west, and comprises the whole north-eastern portion of the State of Nevada. The face of this plateau, which is elevated about five thousand feet above the level of the sea, is covered by broken and detached ranges of mountains, whose highest peaks rise to an altitude of ten thousand feet. Owing to the broken nature of these ranges, the valleys lying between them are connected with one another by low passes, generally forming good natural roads, and furnishing easy grades for the construction of railways. Although numerous streams rise in the mountains, some of sufficient volume for driving mills, yet they all sink and disappear on reaching the sandy valleys below. North-west of this region lies the Humboldt valley, which contains a large amount of arable land.

On the hills and mountains a small variety of pine grows abundantly, but the valleys are entirely destitute of timber, and only covered with sand and a small shrub known as "sage brush." Many of them, in close proximity to the mines, contain vast deposits of nearly pure salt, and, consequently, the price of this substance may be considered as merely nominal throughout the whole country. This district lies between longitude one hundred and fourteen and one hundred and eighteen degrees west, and latitude thirty-seven and forty-one degrees north.

At some period, generally subsequent to the cretaceous epoch, igneous agencies have been active, and probably three-fifths of the whole country are covered with rocks of volcanic origin. The other portions of the surface, where exposed, are composed of granite, syenite, gneiss, slate, and limestone; the two latter being frequently fossiliferous. Triassic fossils occur in many localities, and have been clearly identified; others much older, probably Silurian, have been found near Austin. The country has, however, been too recently discovered to admit of any but imperfect geological explorations. The great majority of the metalliferous veins are in granite, syenite, and slate; the limestone enclosing but comparatively few. Nearly all of the valuable metals are found in this region, such as gold, silver, copper, lead, iron, zinc, antimony, &c. At present, owing to the high price of transport, gold and silver mines are the only ones that can yield profitable results.

Reese River District.—This district is situated on the western slope of the Toiyabe range of mountains, which extends north and south for the distance of one hundred miles, and occupies nearly the geographical centre of the State of Nevada. A number of subordinate districts are situated on this range, and the majority of the mines worked in the country are found on its eastern and western slopes. In this region silver was first discovered, and here the mines have been more fully opened, and a greater number of mills erected, than in any other locality. Austin, the largest town in this part of the country, containing three or four thousand inhabitants, is built in the centre of the district. Nearly all the ores met with, at the surface, consist of chloride, iodide, and bromide of silver associated with native silver; whilst below the water level, ruby silver, stephanite, polybasite, antimonial silver, argentiferous galena, silver fahlerz, xanthocone, &c., besides a great variety of combinations more interesting to the mineralogist than valuable to the miner are found.

The first attempts at working these ores were unsuccessful; but since the introduction into the country of the process of roasting with salt in reverberatory furnaces, the results obtained have been satisfactory, although dry crushing and working the furnaces by hand render the process very costly in a country where labour costs at least four dollars per day. Sixty dollars per ton of 2,000 lbs. is the rate now charged at the *custom mills*, a return of eighty per cent. of the fire assay of the ore being guaranteed.

This district, though small in extent, six miles north and south, by one and a half east and west, contains a remarkably large number of silver-bearing veins. Over five thousand different branches have been discovered and recorded in the district, nearly all of which are visible at the surface; and ninety-five per cent. of this number contain silver, though frequently not in paying quantities. The vein-systems are remarkably numerous, besides which a complicated series of faults renders their study interesting. Six apparently distinct systems have been observed, and the relative ages of four of them satisfactorily determined. The oldest and by far the most numerous veins have a strike north-west and south-east, magnetic, the variation being sixteen degrees east, with a dip of forty-five degrees to the north-east; many of the richest veins belong to this system. The second system has a strike north seventy degrees west, and south seventy degrees east, with a dip of fifty degrees to the north. The third system has a strike of north twenty-five degrees west, with a dip of seventy degrees to the west. These three systems have all well-defined walls, accompanied by slickensides and flucan. The veins of the fourth system strike north and south, and stand perpendicularly. Two other systems are believed to exist, one with a strike east and west, and a dip of forty degrees to the north; the other running north-west and south-east, and dipping thirty degrees to the south-west; the relative ages of these have not been determined.

The mineral belt extends the whole length of the district, by nearly two miles in width. All of the richest mines are in this belt, which runs nearly parallel with the Toiyabe mountains. Owing to the recent discovery of these mines, and the disadvantages incident to new countries, no deep or extensive explorations have yet been made; about four hundred feet being the greatest depth yet attained by any incline following the dip of the veins. Many of the mining operations have been conducted with a very moderate degree of skill, and are consequently of little value. The following description embraces a few characteristic mines on each well-established system of veins, commencing with the oldest.

FIRST SYSTEM.—The *Oregon*, like all the veins of this system, has a strike north-west and south-east, and a dip of forty-five degrees to the north-east; its average thickness is about one foot. This was one of the first veins opened in the country, and has yielded a considerable amount of ore; the average produce of the rock worked has been over a hundred dollars per ton. An incline has been sunk to the depth of four hundred feet, and drifts run, on each side, for over a hundred feet. This vein was irregular and broken near the surface, but for the last hundred feet appears to be more regular. It contains the ores usually found in the district, such as chloride of silver, ruby silver, and stephanite, associated with large quantities of manganese.

North Star.—This mine has been opened to the depth of three hundred and forty feet, and drifts driven to the eastward for two hundred feet. Its average thickness is about eighteen inches, though in some portions it is three feet in width: the composition and yield of the ores are similar to those of the Oregon.

Revenue.—The thickness of this vein between the walls is two feet; the ore is found in streaks of from three to ten inches. The ores in this vein are rich, the yield by assay of the pure mineral having been over five hundred dollars per ton, whilst the yield by working in the mill has not been above two hundred. Larger pieces of native silver have been taken from this mine than from any other on Reese River.

North River.—This vein, so far as it has been followed down, has increased in thickness; at the surface it was only six inches wide, but at the depth of a hundred feet it is over a foot in thickness, and in some portions even eighteen inches. The ore in this mine resembles that of the Revenue. The yield of the ore worked has been about two hundred dollars per ton; nearly all of the veins of this system are double, consisting of two or more foliations, usually from two to four feet apart.

SECOND SYSTEM.—The strike of these veins is north seventy-five west, and south seventy-five east, and the dip fifty to the north. These veins are usually large; from two to eight feet thick.

Whitlatch Union.—This is one of the largest veins in the district, being from four to eight feet thick; it has been opened to the depth of a hundred and twelve feet, and drifts driven on each side for one hundred feet. The ores so far found belong to the surface class, and their yield has been about a hundred dollars per ton. This vein was divided by a fault, and much labour expended before its continuation was again found.

Savage.—This mine has been more fully opened than any other in the district, and hence it has produced more ore. An incline has been sunk to the depth of three hundred feet, and two levels opened; the lower one for a distance of six hundred feet, and the upper for about four hundred. The ores are antimonial with small amounts of ruby silver.

THIRD SYSTEM.—The strike of these veins is north twenty-five degrees west, and south twenty-five east, and their dip seventy degrees to the westward. All their ore is cased in greenstone with a large amount of clay, and they usually carry a considerable body of water. The ore is not so generally diffused throughout the veins as in the two older systems, but is found in separate deposits, and the thickness of the vein is not so regular.

Whitlatch Yankee Blade.—The thickness of this vein is from two to four feet; it has been opened to the depth of three hundred feet, and drifts driven to the distance of seventy feet. The ores are similar to those of the older systems, except that they contain larger amounts of arsenic; the average yield has been about a hundred dollars, though a number of tons have produced three hundred dollars per ton.

The Confidence is almost a fac-simile of the Whitlatch Yankee Blade.

FOURTH SYSTEM.—These veins strike north and south, and stand perpendicularly; they are less metalliferous than those of the older systems, and are more spotted, many portions being entirely barren. They all, however, contain gold in sufficient amounts to constitute a portion of the total value of the ore. So far as is known, this is the latest, or one of the latest of the known systems in the country; consequently it has been less disturbed by faults than any other. One of the veins, which has been opened to the depth of a hundred feet, has a thickness of about two feet. The average yield of its ores is not known, but some samples have afforded by assay a hundred and fifty dollars per ton. The average yield of the sulphides taken from below the water level, was nearly double that of surface ores.

Amador District.—This district adjoins that of Reese River on the north, and is on the same mineral belt. The country rock is slate, and the veins, though large, are not so rich or numerous as in Reese River.

No veins have been found passing from the granite into the slate, and none between the two formations. The usual strike is north-west and south-east, and the dip about forty degrees to the north-east. Many of the veins show evidence of great dynamic action, their walls being often polished as smooth as glass. The Amador, Rough and Ready, and Corral Mines have been opened to depths varying from two to four hundred feet; the average thickness of the veins being from four to six feet. These veins are capable of producing a large amount of ore, but, so far, have not proved sufficiently rich to leave a profit to the mine-owners after paying sixty dollars per ton for reduction.

Ravenswood District.—This district lies about eighteen miles north-west from Austin; the country rock is slate, and the veins are very numerous and rich in copper and lead, but rarely yield over forty dollars per ton in silver, which will not, at the present scale of charges, pay the expenses of working.

Cortez District.—Cortez is sixty miles north of Austin, and but a short distance from the Humboldt River. Wood, water, and grass are comparatively abundant; the country rock is slate, granite, and limestone. There are many known veins in this district, the principal being the Nevada Giant.

Nevada Giant.—This crops to the surface for the distance of two and a half miles, its thickness being from fifty to two hundred feet, with a direction nearly north and south, and dipping to the east. The vein contains many deposits or chimneys showing at the surface large quantities of valuable ore, which principally consists of disseminated stephanite.

Washington District.—Forty miles south of Austin is Washington, which contains a large number of veins, yielding argentiferous galena: they are generally rich in lead, and assay from twenty to sixty ounces of silver to the ton of ore; the country rock is slate and limestone.

Union District.—This district is sixty miles south-west from Austin on the western slope of the Shoshone range of mountains, which are generally well covered with wood. The enclosing rock is syenite, the veins are small, rich, and numerous; in the slate they are larger, less productive, and fewer in number, while the value of the few found in limestone has been in no way determined. Only one vein in this district has been opened below the water level, and in that, sulphide of silver was found. Hence all the ores worked in this neighbourhood have been chlorides. The general direction is north-west and south-east, with a dip of forty degrees to the north-east.

The Pleiades.—This mine has been opened to the depth of two hundred feet, and drifts driven on each side for a distance of seventy or eighty feet; the thickness of the vein varies from two to four feet; the ores are similar to those found near Austin, but contain rather more lead; their average yield has been nearly one hundred dollars per ton. There are many other similar veins in the district, such as the Clipper, Idaho, Shaw, Silver Moon, &c.; some being rich in gold, as the Franklin and Shamrock.

The Great Eastern.—The strike of this vein is north and south, with a dip of about sixty degrees to the west. It is composed of a number of different bands, varying from ten to one hundred feet in thickness, with an aggregate width of from three to four hundred feet. It crops to the surface for a distance of over ten miles, and passes through different country rocks, such as slate, limestone, and porphyry, but it is only productive in the latter. In the Great Eastern claim a tunnel has been run through two of the strata, one thirty-five feet thick, and the other about ten. The larger stratum, cut by the tunnel, afforded assays of forty dollars per ton; the smaller one yielded some samples producing a hundred dollars per ton.

Smoky Valley District is situated seven miles south-east from Austin, on the eastern slope of the Toiyabe range of mountains; the country rock is slate and granite; many small rich veins are found in the granite, but the principal lodes are in the slate. Those which have a direction east and west, and a dip of about forty degrees north, are known as the Mammoth and Smoky Valley, have a thickness of from twenty to forty feet, and run parallel, about one hundred yards apart, for a distance of over three miles. An incline has been sunk on the Smoky Valley, to the depth of two hundred feet, but the average yield has not yet been determined.

The Twin River District is situated on the eastern slope of the Toiyabe mountains, about fifty miles south from Austin; the country rock is slate and granite; all of the valuable veins are found in the slate, or at the point of junction of the slate and granite. They are large and numerous, many of them having a thickness of from twenty to forty feet; their general direction is nearly north and south, with a dip of seventy degrees to the west.

The Murphy is the only vein that has been opened and worked sufficiently to prove its value; it is twenty feet thick, with a paying streak of from five to seven feet in thickness. Many other veins are of equal or greater size, and present equally favourable indications.

The Vanderbilt is nearly seventy feet in thickness, with a foot wall of granite, and a hanging wall of slate, and can be traced on the surface for a distance of over a mile. The ravines cut the veins at right angles, to the depth of nearly two thousand feet; thus affording great facilities for opening mines by means of adits driven into the vein on its direction. This district has been recently discovered, but bids fair to become an important one; wood, water, and salt are abundant.

Mammoth District is seventy miles south-west from Austin; the veins in this locality are very numerous, regular, and uniform in direction, dip, and thickness; being but little disturbed by faults. Their general strike is north-west and south-east, and their dip forty degrees to the north-east; the average thickness is about two feet. None of the veins have been opened to any considerable depth, and all the ores, so far worked, have been from the surface. A large amount of ore can be taken from the backs, that will yield about forty dollars per ton. This district has not been discovered sufficiently long to admit of any but superficial openings being made, but it promises to become an important mining country.

San Antonio District.—This district is situated about a hundred miles south from Austin. It is an almost perfect counterpart of the Reese River district. The vein systems are equally numerous, but their relative ages have not been ascertained. The surface ores have the same composition, and are of about equal richness, although the veins are larger; the country rock is generally slate. No water is found within a mile and a half of the mines, and then only in small quantities; nine miles from the mines, water is met with in sufficient abundance to supply a mill, and wood can be procured within six miles of the water.

The Liberty.—This mine has been opened to the depth of two hundred and fifty feet without finding water. The thickness of the vein is from four to eight feet; the run is west-north-west and east-south-east, and the dip forty-two degrees to the north. So far as yet opened on, the ores are chlorides and iodides; more horn silver is found in this mine than in any other in the country. The yield of the ores, selected for working, has been over three hundred dollars per ton.

Las Animas.—This mine has not been opened as fully as the Liberty, which it very much resembles, only that the vein is thicker. Want of wood and water has greatly retarded the development of this district.

Silver Peak is one hundred and forty miles south from Austin. A large amount of exfoliated ore, which has fallen from the backs of the veins, is found at the surface; some of these ores have been worked, and good results obtained: the country rock is granite, slate, and limestone. Salt is abundant, and wood and water moderately plentiful. In addition to the foregoing two great mining regions of Nevada, there are numerous others which are as yet of minor importance, but which are being steadily and rapidly developed. It would, however, be impossible to enumerate even the names and positions of the different veins which have been discovered in this State.

Other Mining Districts.—In Esmeralda County, where there are some important veins, there are three distinct hills in which the principal mines are situated.

Silver Hill, in which are the Esmeralda ledge; the Winnemucca, Falls of the Clyde, St. Louis, Utah, Antelope, Red White and Blue, Cedar, Greenback, Lily of the West, Locomotive, and many others, on some of which, shafts have been sunk 300 feet, and rich ore taken out; about twenty levels have also been run into the hill, some of them from 500 to 600 feet in length: no water of any consequence has been met with. Middle Hill contains many promising veins, but sufficient work has not been done on them, to determine their true characters. Last Chance Hill, in which is the Wide West ledge, the Del Monte, Golden Age, Empire, Crocket, Ætna, and many others. Considerable amounts of the precious metals have been taken from them, but during the last twelve months the same amount of work has not been done on mines in this district, as in the previous year, owing to litigation and insecurity of title. In this district many hundreds of tons of boulders from the surface have been broken by miners and worked in the mills, generally paying well. Aurora is the county seat. During the past year, mines in the Bodie District, about twelve miles west from Aurora, have been worked with profitable results. Two large mills have been erected in the district, and are working ores which pay from $50 to $100 per ton. Montgomery, Blind Springs and Hot Springs Districts, about forty miles south-east from Aurora, have also been discovered within the last two years. Lake District, near Walker Lake, has also been recently discovered. Many large veins, with prominent croppings, are being worked, and some ore rich in both gold and silver has been taken out. Columbus, Silver Peak, Red Mountain, Cottonwood, and Minnesota Districts, about seventy-five miles south-east from Aurora, have also been discovered during the past two years. The veins are large and well defined, being from one to four feet wide, and can be traced for several hundred feet, the croppings being generally prominent.

The Montezuma Mine, in Humboldt County, is situated in Trinity District, fifteen miles west of Unionville. The Montezuma ledge runs in an east and west direction, and dips north at an angle of seventy degrees. The principal country rock is a metamorphic slate. The vein is ten feet thick, well defined, and has a clay selvage on each side. The ore principally consists of oxide of antimony, carbonate and oxide of lead, and arsenical pyrites. So far as yet explored, this vein contains but little refuse. The ore yields, by mill process, without roasting, an average of seventy-five dollars of silver per ton. On this vein two shafts have been sunk, each thirty feet in depth, and drifts have been extended to an aggregate length of eighty feet. The distance from the mine to the Humboldt River where at a moderate cost good water-power can be obtained, is four miles, over a good natural road. The work done, thus far, has been by way of exploration only.

The Montana Mine is situated n the Sacramento District, eighteen miles south of Unionville. The lode is in slate and limestone. It runs north and south, and pitches west seventy degrees.

The Sheba Mine is near Star City. Its ore principally consists of sulphide of antimony, brittle silver ore, and antimonial silver, and is found in deposits which are distributed irregularly through a stratified belt, or ore channel, about a hundred and fifty feet thick, which lies between quartzite and slate. In the development and working of this mine, three thousand feet of levels and drifts have been run at a cost of $75,000. The aggregate yield has been about $70,000; and the average produce per ton, $140.

The De Soto Mine is situated immediately south of the Sheba. The vein is four feet thick, runs in a northerly direction, and has clay selvages and polished walls; the country rock is slate. Upwards of a thousand feet of tunnel have been run. About two hundred tons of ore have been taken out, which averaged nearly $100. The ore closely resembles that of the Sheba.

The Yosemite Mine is situated two miles south-east of Dun Glen. The formation in its vicinity is slate and limestone. The vein is two feet thick, and has clay selvages and striated polished walls. It runs north and south, and stands almost vertically; a shaft has been sunk to the depth of fifty feet, and from this about fifty feet of drifts have been run. Three hundred feet of a working tunnel have also been run. Several small faults occur in the vein, which give it a broken appearance. The ore is a black sulphide of silver, containing a small proportion of carbonate of copper. It is exceedingly rich, some of it assaying as high as $13,000 per ton.

The Gem of the Sierras is situated in Sierra District, five miles from Dun Glen, in a limestone formation. The vein is twenty inches in thickness, runs in an easterly and westerly direction, and dips 70° south. A considerable portion of the gangue is calcareous spar. The ore consists of sulphide and chloride of silver, mixed with a little carbonate of copper; and the parcels worked have yielded at the rate of $175 per ton.

The Tallulah Mine lies a mile and a half west of Dun Glen, and comprises several ledges running north and south, and dipping west. The enclosing rocks are slate, syenite, and porphyry. A level has been extended five hundred feet, cutting at an average depth of a hundred and eighty feet, a well-defined vein separated from the enclosing rock by selvages and polished walls. The ores chiefly consist of antimonia, sulphide of silver, with a little native metal, and after selection yield about $150 per ton.

The Pride of the Mountain is situated in the Winnemucca District, twenty miles north of Dun Glen. The vein runs north and south, pitches east at an angle of 45°, and is about fourteen inches thick. It is well defined and continuous, and has clay selvages and striated walls. The country rock consists of a soft metamorphic slate, and much of the gangue is calcareous. In making the incline, forty tons of ore were taken out which, without selection, yield $80 per ton by the pan process.

The Manitowoc Mine lies two miles south of Unionville, in Buena Vista District. The vein is two feet thick, runs north and south, and dips west at an angle of 45°. The country rock is slate. The ore consists of sulphide of silver and xanthocone containing a small proportion of copper, and is easily worked by the common pan process.

Devil's Gate District is situated in the north-west portion of Lyon County, and is the oldest district in it. The lodes in the most northerly portion are auriferous in their character, many being well defined. The most prominent veins, however, are the Wide West Twin, and Buckeye. Lying south are the Daney, and Charles Cany Companies. The Daney Company has erected permanent works, and ore has been met with in several places in the mine, but not sufficient in amount to keep in operation the ten-stamp mill which the Company has erected. There are 34 mills in this county, principally employed in the reduction of ore from the Comstock vein.

There are several veins in Nye County, which in the aggregate produced during the year 1865 about $100,000 worth of bullion; this principally consisted of silver yielding only a small proportion of gold.

In Washoe County there are no mines worked to any extent, but it contains several reduction works, principally occupied in the amalgamation of Comstock ores.

According to the Report of the State Mineralogist, there are now in the State of Nevada no less than 116 distinct reduction works, provided in the aggregate with above 2,200 stamp-heads.

CHAPTER XV.

CENTRAL AMERICA AND SOUTH AMERICA.

GUATEMALA — MINES OF THE CENTRAL AMERICAN MINING COMPANY — MINING DISTRICT OF ALOTEPEC — PRODUCE OF SILVER—SOUTH AMERICA — MINES OF PERU—ANNUAL PRODUCTION OF SILVER—BOLIVIA—MINES OF POTOSI—THEIR PRODUCTION OF SILVER—CHILI—MINES OF COQUIMBO—ANNUAL PRODUCTION OF CHILI—NEW GRANADA—SANTA ANA MINES—TABLE SHOWING PRODUCTION OF SILVER IN VARIOUS COUNTRIES.

CENTRAL AMERICA.—Comparatively little is known of the geology of this country, except that the predominating rocks are granite, gneiss, and mica slate, and that the abundance of igneous rock bears witness to extensive volcanic action. Silver mines are situated in various parts of Central America, but the only ones of which we have any authentic information are the following:—

Mines of the Central American Mining Company.—The mining district of Alotepec is situated in about 14° 40′ of north latitude, and 88° 58′ of west longitude; being a portion of the Department of Chiquimula in the Republic of Guatemala.

The mountain of Alotepec forms part of the great chain which runs towards Honduras, in which the best silver veins of Central America are found. As a mining district it appears to have been known more than a hundred years ago, and a period of great prosperity occurred about seventy years since. The veins are found in porphyritic rocks, and occasionally pass into sandstones and limestones, which appear to have been uplifted by the porphyry. Their general direction is from east to west in the sandstone and limestone formations, and south-east and north-west in the porphyry. The mineral wealth of the district principally consists in silver, lead, and copper. The silver is found both with lead and copper; also as a sulphide of silver, and in the state of chloride. Lead is found as sulphide and carbonate; the copper is in the form of pyrites, green and blue carbonate, and as an oxide. There is also a great abundance of iron ore in different parts of the mountains. The rocks supply abundance of lime for mortar, building-stone of various kinds, clay for bricks and tiles, and an inferior kind of sand is found in the ravines. The firestone in the neighbourhood is not good, but it may be obtained from Tuitiapa, and fireclay at Esquipulas and near the city of Guatemala.

The number of veins and mineral deposits in the mountains of Alotepec cannot be less than thirty-eight or forty, which the late Mr. Floresi divided into five groups or divisions, viz.:—

1st. The mines of Don Manuel Midense.
2nd. Those of Don Cruz Duarte.
3rd. San José, San Rafael, and Santa Rita.
4th. Anderson's Hope and El Tajo.
5th. Taylor's Hope, &c. &c. &c.

The first group comprises the mines of San Pantaleon, Santa Rosalia, Socorro, and Santa Fé. These mines are on the north side of the mountain, looking towards the Rio del Valle, and are surrounded by a fine forest, chiefly of pine and oak. The pasture land enclosed is sufficient for about 180 animals during the greater portion of the year. There is a supply of water sufficient for stamps, &c.; and about two and a half miles from the mines, is a piece of land called "La Vega de San José," having the advantage of extensive water rights, and on which very convenient reduction works have been constructed; the German barrel process being that adopted. Before their purchase by the present Company, the mines had been worked at different periods, but very unskilfully. The vein of San Pantaleon is that which has yielded the best produce. The veins of Santa Rosalia, Socorro, and Santa Fé have been worked at different times, and produced moderate quantities of ore of good *ley*. They are well situated for deep cross-cuts, and can be worked and drained without the necessity of a shaft, requiring only such openings from the surface as may occasionally be wanted for ventilation.

To the second group belong San Carlos, Rosario, San Miguel, San José de Atutilca, and other mines. The veins, although in general not very wide, are promising; the ores are rather mixed, and principally consist of compounds of copper, lead, sulphur, and silver, generally better adapted for smelting than amalgamation. The mines of this group are also well situated for adits, and are not more than a league from the Vega de San José. In the third group are the mines of San José, San Rafael, Santa Rita, and San Domingo. San Rafael and San José appear to have been worked extensively, but they do not present such facilities for deeper levels as the other mines. The fourth group comprises the mines of Santa Catalina or Anderson's Hope, and El Tajo. Mr. Floresi considered them to be neither regular lodes, nor beds, but a reunion of several small veins. The ores of El Tajo form an excellent flux in the smelting of other ores. The fifth group consists of Taylor's Hope and adjoining veins. Taylor's Hope is about six miles from the reduction works.

The rivers in the immediate neighbourhood of the mines allow of the reduction establishment being worked entirely by water-power; the forests are capable of supplying charcoal and wood to any extent, and very few places can furnish a larger variety or a greater abundance of timber than this district.

The silver produced by the Central American Mining Company has been chiefly obtained from the mine of San Pantaleon, but recently, considerable quantities have been raised at San Carlos. Of late years also, some ore has been extracted from San José de Atutilca.

It will be observed by reference to the annexed tabular statement that the production of these mines has considerably fallen off since 1863, but discoveries are expected to result from the exploratory works now in progress, and it is hoped that the returns for the present year may again increase.

Statement of Silver produced by the Central American Mining Company, from 1858 to 1865.*

		oz.	oz.
1858	Produce of ore sold . . .		18,187·80
1859	" " . . .		62,623·45
1860	" " . . .		67,825·95
1861	" " . . .	45,508·05	
	" of bar silver sold .	80,579·94	
			126,087·99
1862	" of ore sold . . .	34,662·45	
	" of bar silver sold .	21,686·19	
	" " " .	65,897·00	
			122,245·64
1863	" of ore sold . . .	4,955·25	
	" of bar silver sold .	12,795·84	
	" " " .	109,446·29	
			127,197·38
1864	" of ore sold . . .	2,923·95	
	" of bar silver sold .	64,762·96	
			67,686·91
1865	" of ore sold . . .	1,768·20	
	" of bar silver sold .	27,354·58	
			29,122 78
	Total produce		620,977·90 oz.

South America.—The silver mines of that portion of South America which constitutes the republics of Peru, Bolivia, and Chili, are next in importance to those of Mexico and Nevada, and would, had they been more favourably situated, probably be capable of affording a larger annual amount of silver than even the mines of Mexico. The great elevation of many of the argentiferous districts of South America is, however, a great drawback to their value, since nothing but the extreme richness, or the great abundance of the ores, could afford a sufficient inducement to undertake the working of mines in such desolate and inhospitable regions.

Peru.—The mines of the Cerro de Pasco are the most celebrated of Peru; the principal ores worked being of the description known as *pacos*, which are analogous to the colorados of the Mexican miners, and consist of ferruginous earths containing varying amounts of silver.

In order to form an idea of the enormous quantities of these argentiferous gossans, which nature has deposited in the calcareous hills of this district, it may be stated that the bed of silver-bearing oxide of iron of Pasco or Yauricocha, had been worked, without interruption, since about the commencement of the seventeenth century; and that although, during the first twenty years previous to the commencement of the nineteenth, they had produced above five million marks of silver, very few of the workings then penetrated to a depth of above a hundred feet below the surface.

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* Furnished by Mr. J. Phillips.

The metalliferous deposit of Yauricocha crops out on the surface for a length of 2,500 fathoms, and a width of 1,200 fathoms. M. de Rivero considers the pacos of Santa Rosa, one of the most productive mines in the district, to be a deposit and not a true vein, since they run parallel with the general formation of the country, and the gangues have no crystalline or drusy structure.[*]

These mines were accidentally discovered by an Indian in 1630, and were, for a long period, regarded as the richest in any portion of the American continent. They have, however, been badly worked, and, many years since, one of them fell in upon the workmen, of whom three hundred were killed, from which circumstance the mine has acquired the name of *Matagente*, or "Kill-people." The town of Cerro de Pasco is 13,673 feet above the sea, and when the condition of the mines is prosperous, sometimes contains 18,000 inhabitants; according to Tschudi, the two principal argentiferous veins of this district are the Veta de Colquirirca and the Veta de Pariarirca. The former of these runs in nearly a straight line from north to south, and has been traced for a length of 9,600, and over a breadth of above 400 feet. The second takes a direction from east-south-east to west-north-west, and is supposed to intersect the other under the market-place of the city. Its known extent, in length, is 6,400 feet, and its width 380 feet. Steam-power was first introduced into these mines in 1814, by the celebrated Richard Trevithick, but the acid waters of the veins acted so rapidly on the pumps, that they were very quickly destroyed, and in 1832 only one of the engines remained at work.

In addition to the district of Cerro de Pasco there are various others in Peru which have produced large quantities of silver. The principal of these are those of Caxamarca, Pataz, Huamanchuco, and Hualgayoc. In the Cerro de San Fernando alone, included in the latter district, there were in 1840 no less than 1,700 *bocaminas*, or mine openings. There are also numerous silver mines in the southern districts, but the amount of metal produced is small in comparison with the richness and extent of the veins.

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[*] Ann. des Mines (3), f. ii. 169.

The total amount of the silver produced from the principal mines of the Cerro de Pasco from 1784 to 1827, was 4,962,929 lbs. troy.

The total annual production of the silver mines of Peru is estimated at 299,000 lbs. troy.

BOLIVIA.—The mines of Potosi, which once formed a portion of the viceroyalty of Buenos Ayres, are now included in the Republic of Bolivia, and have, since their discovery in 1545, yielded almost fabulous amounts of silver. In this locality thirty-two principal veins, besides numerous smaller ones, have been worked in an isolated mountain called *Hatun-Potocsi*, or Great Potosi, the summit of which reaches an elevation of 16,000 feet above the level of the sea. From the time of the discovery of these mines, up to 1571, when the process of amalgamation was introduced, the ores were treated exclusively by fusion. The Spanish conquerors of the country, being military men, were but little skilled in metallurgical operations; and having been unable to effect the fusion of the mineral by the aid of bellows, adopted the primitive process employed by the natives for the treatment of ores obtained from the neighbouring mine of Porco, which had been in operation long previous to the conquest. For this purpose, according to Humboldt, small portable furnaces, called *huayres* or *guayres* were established on the mountains in the neighbourhood of Potosi, wherever they could be fully exposed to the action of the prevailing winds. These furnaces consisted of cylindrical vessels of refractory clay, pierced with numerous holes, in which the silver ores were introduced, together with galena and charcoal, in separate layers. The air, which entered the holes before referred to, caused the fuel to burn with great intensity; and when the combustion became too active, and the consumption of charcoal consequently very rapid, the furnace was removed to some locality less exposed to the wind. Early travellers who visited the country, speak with enthusiasm of the effects produced by these fires, more than six thousand in number, which, every night, lit up the mountains in the neighbourhood of the mines. The galena necessary for the operation of smelting was obtained from a neighbouring mountain called *Huayna-Potocsi.*

The argentiferous products obtained from the portable furnaces were re-melted in a fire kept up by means of copper blow-pipes, from six to seven feet in length, and of which from ten to twelve, used by as many different persons, were employed at the same time.

It will be readily conceived that the loss of silver experienced by this primitive process must have been exceedingly great.

The period of the greatest productiveness of the Potosi mines was the century which immediately followed their discovery, their average annual yield from 1545 to 1556 having been about $11,600,000. Shortly after the commencement of the seventeenth century their production began to decline, and at its close their annual yield had receded to between three and four millions of dollars. The annual yield of the Potosi mines was estimated by Chevalier, in 1845, at from 48,000 to 60,000 lbs. troy. Although the produce of the ore has considerably decreased, in proportion as greater depths have been attained, the mines of Potosi are far from being exhausted, and there can be no doubt but that, if intelligently worked on a large scale, under the protection of a reliable Government, they might be, at least to a great extent, brought back to their former state of prosperity and productiveness. According to Whitney there were in 1852, in the province of Potosi, above 1,800 abandoned silver mines, and only 26 at work; and in the remaining mining districts of the country there were no less than 2,365 abandoned mines, and only 40 in course of working.

The total amount of silver produced by the mines of Peru and Bolivia, from the earliest period up to 1845, was estimated by Chevalier at 155,839,180 lbs. troy.

Chili.—Domeyko, who has described the mining districts of Chili, divides the rock formations of that country into three distinct groups, viz *

1st. Secondary stratified, prior to the upheaval of the Andes.

2nd. Igneous eruptive masses, of the period of the upheaval of that chain.

3rd. Tertiary beds, posterior to that epoch; generally speaking, the veins affording gold and copper belong to the second group, whilst those of argentiferous copper, containing these metals in combination with sulphur, arsenic, and antimony, are usually included in the first. The gold veins are principally enclosed in granite, and those of copper, uncombined with silver, arsenic, or antimony, are chiefly found in diorites, porphyries, and eurites, or in some other eruptive igneous rock. Chloride of silver and the native amalgams are, in most

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* Ann. des Mines (4), t. ix. 22.

instances, met with near the point of junction of rocks belonging to the first and second classes.

The mining districts of Chili may be divided as follows:—

Mountains north of the valley of Huasco—This is the richest silver district of Chili, but also contains valuable mines of copper and gold. The most productive copper mines of this section are those of Carrisal.

Districts lying between Huasco and Coquimbo—In this region are the rich groups of copper veins of San Juan and La Higuera, which annually supply large quantities of copper ores of a high percentage produce. The mines from which these are obtained, are situated in dioritic rocks. On a line between Arqueros and Agua Amarga, which represents the course of junction of the first and second groups, there are numerous veins affording metallic silver, chloride of silver, and native amalgam.

The third district lies between the valleys of Coquimbo and Aconcagua, where the granite extends far inland, and the gold-bearing veins present a greater development. The whole of the granitic district is more or less auriferous, and on its borders are numerous veins affording copper ores.

The fourth district lies to the north of Aconcagua, where, as in the foregoing, the granite is traversed by auriferous veins; and mines of silver and argentiferous copper ores are worked above the level of the various ravines existing in the stratified rocks forming the elevated chain of the Andes.

The usual gangues of the copper veins are quartz and hornblende; whilst carbonate of lime and sulphate of baryta accompany the ores of silver. Gold is commonly associated with quartz and sulphide of iron. The most abundant ore of silver is the chloride, together with bromide of silver and the native metal. The chlorides are found in the usual ferruginous earthy deposits, called pacos and colorados by the miners of South America. In addition to these, there is a great variety of sulphides and arsenides of silver. The ores usually contain from 100 to 250 oz. of silver per ton; but it is a remarkable fact that the blendes and galenas occurring in the rich silver regions of Chili, as a rule, scarcely contain a trace of that metal.

The most important silver mines of Chili are those situated in the neighbourhood of Copiapo. All that portion of the country lying above the parallel of Valparaiso is rocky and sterile, excepting three tongues, or narrow bands, varying from three-fourths of a mile to one mile in width, extending inwards from Coquimbo, Huasco, and

Caldera. All the remainder is a desert, and in it, at distances of from thirty to forty leagues from each other, are the various mining districts. The mines of Coquimbo and Huasco chiefly afford copper, whilst those of Copiapo are rich in silver.

The principal workings are those in the vicinity of Chañarcillo and Tres Puntas; the first, sixteen leagues south, and the second, thirty to the north-north-east of Copiapo. The mines of the former locality were discovered, in 1832, by a muleteer, and were worked with large but gradually diminishing returns until 1836, when the veins were found to have been cut off by a stratum of tough limestone, called in that country a *mesa*, or table. On this discovery being made, the mine proprietors of the district became generally discouraged; but one of their number, more enterprising than the others, having sunk to a depth of 266 feet through the unproductive rock, found on the other side a rich deposit of silver ore. Several of these barren strata have since been intersected by sinking, and it has been invariably found that the veins are rich between them, and that the largest accumulations of ore are met with near the planes of contact of the limestone with the adjoining rocks. The first mine discovered in this district was that known as the Descubridora. The three principal mines of the Tres Puntas are La Buena Esperanza, La Salvadora, and the Al Fin Hallada; whilst in the district are some twenty other more or less productive and profitable workings, together with a great number producing little or nothing.

The development of the mineral resources of Chili has been more recent than that of the other South American States; but its comparatively flourishing political situation, and the internal quiet which it so long enjoyed, have enabled the workings to be established on an extensive scale, and has, within the last few years, led to a considerable increase in the amount of silver annually produced. Under the Spanish dominion, the production of silver was inconsiderable, and in 1800 the annual yield was estimated by Humboldt at only 13,300 lbs. troy. Since the discovery of the rich mines of Copiapo, however, in 1832, the production of silver has very much increased. The total yield of silver, up to 1810, was estimated by Chevalier at 804,000 lbs. troy, and from 1804 to 1845 at 1,803,636 lbs.

The silver exported for the eight years from 1834 to 1841, both inclusive, was, according to the returns of the British consuls, based on the confessedly defective information furnished by the Chilian Government, as follows:—

	Marks.
1834	164,935
1835	231,988
1836	163,158
1837	219,482
1838	135,854
Carried forward	915,417
Brought forward	915,417
1839	148,089
1840	141,621
1841	140,123
Total	1,345,250

Mr. Danson estimates the total exports of silver from 1804 to 1848 at $38,555,205.* Whitney considers that the production of the country from 1846 to 1853 was probably about 1,750,000 lbs.; making the entire yield of the mines of Chili, up to that time, 4,357,656 lbs. troy. We are without any authentic returns of the produce of the Chilian mines since that period.

NEW GRANADA.—*The Santa Ana Mines* are situated in the Province of Mariquita, State of New Granada. They contain deposits of various argentiferous ores, chiefly consisting of silver-bearing pyrites mixed with native silver, ruby silver, and various sulphides. This locality also affords argentiferous galenas and blendes. The ores are enriched by a process of stamping and dressing previous to being submitted to barrel amalgamation. The mines have been worked to a depth of 120 fathoms, and the quantity of ore raised in 1864 amounted to 1,570 tons; which, after being reduced by concentration to 453 tons, yielded 78,281 ounces of silver. The cost of grinding and amalgamation, in 1861, amounted to $43·63 per ton; the loss of mercury being 5·72 lbs. per ton of concentrated ore treated. The loss of silver was stated in that year to have been only 5·33 per cent.†

The following is a statement of the silver produced from the Santa Ana Mines, from 1852 to 1864, both inclusive:—

Year.	oz.
1852	57,169
1853	31,403
1854	55,009
1855	84,415
1856	129,389
1857	158,519
1858	160,036
Carried forward	675,940
Brought forward	675,940
1859	140,509
1860	84,771
1861	81,044
1862	93,436
1863	112,474
1864	78,281
	1,266,455

The following table gives the approximate yields, in pounds troy, of the principal silver-producing countries of the world, at the commencement of the present century, and for the years 1850 and 1865. In cases where the return for the year indicated could not be obtained, the produce for the nearest year for which they could be procured has been substituted. The quoted produce of the various European countries, and of the United States of America, may be taken in each instance as a sufficiently close approximation; but the figures relating to Mexico, Central America, and South America, must be regarded as estimates only. A large proportion of the precious metals produced in those countries is annually exported without passing through the hands of the Government officers, and consequently the most reliable information that can be procured is but little to be depended on. No systematic investigations have been made on the spot by competent persons since the date of the writings of Duport and Chevalier.

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* Quarterly Journal of the Statistical Society of London, March 1851, p. 40.

† It is difficult to understand how the loss of silver should be so small with so large an expenditure of mercury.

TABLE SHOWING APPROXIMATE YIELD OF THE PRINCIPAL SILVER-PRODUCING COUNTRIES.

	1800.		1850.		1865.	
	lbs. troy.	Ratio per cent.	lbs. troy.	Ratio per cent.	lbs. troy.	Ratio per cent.
Russian Empire . .	58,150	2·5	60,000	2·1	58,000	1·5
Scandinavia . . .	141,000	6·0	20,400	0·7	15,000	0·4
Great Britain . .			48,500	1·7	60,500	1·5
Hartz			31,500	1·1	28,000	0·6
Prussia			21,200	0·7	68,000	1·7
Saxony			63,600	2·2	80,000	2·0
Other German States			2,500	0·1	2,500	
Austria			87,000	3·1	92,000	2·2
France			5,000	0·2	18,000	0·4
Italy			...	...	*25,000	0·6
Spain			125,000	4·4	110,000	2·8
Australia, New Zealand, British Columbia, and Nova Scotia	...	...	10,000	0·4	9,500	0·2
Chili	18,300	0·8	238,500	8·4	299,000	7·3
Bolivia	271,300	11·6	130,000	4·6	136,000	3·3
Peru	401,850	17·2	303,150	10·7	299,000	7·4
New Granada . .	5,000	0·2	13,000	0·5	15,000	0·4
Brazil	1,200	...	675	...	1,500	0·4
Mexico	1,440,500	61·7	1,650,000	58·4	1,700,000	42·3
United States . .	...	...	17,400	0·7	1,000,000	25·0
Total	2,337,300	100	2,827,425	100	4,017,000	100

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* Obtained from the island of Sardinia, where it is found associated with galena.

CHAPTER XVI.

TREATMENT OF SILVER ORES BY AMALGAMATION—PATIO PROCESS, &c.

PATIO PROCESS—MAGISTRAL—SALT—MERCURY—LIME—COPPER PRECIPITATE—COMPOSITION OF ORES—ROUGH STAMPING—FINE GRINDING—RASPAR—LOSS OF GOLD—THE PATIO—INCORPORAR—TREADING OF TORTA—WASHING—STRAINING AMALGAM—DISTILLATION—LOSS OF SILVER—LOSS OF MERCURY—CHEMICAL REACTIONS OF THE PATIO—RESULTS OF THE PATIO PROCESS—AT GUANAXUATO—FRESNILLO—REAL DEL MONTE—VIRGINIA CITY—AMALGAMATION BY HOT PROCESS—ESTUFA AMALGAMATION.

It has been found after numerous trials that the ores of silver, with the exception of argentiferous galenas, do not generally admit of mechanical concentration, and they are consequently, after careful selection, in most cases, subjected to metallurgical treatment. The difficulty of treating ores of silver by mechanical means, arises from the fact of the greater portion of this metal being finely disseminated in the veinstone in the form of various brittle sulphides, which, on the pulverisation of the ores, become so finely divided as to float off in suspension in the water employed for concentration. Various carefully conducted experiments made on this subject by Berthier, on samples of silver ores obtained from the different districts of Mexico, go to show that under favourable circumstances, nearly one-half the amount of silver originally present, is lost by even the most careful process of washing that can be applied to them. It must also be borne in mind that, even had the results obtained by mechanical preparation been more favourable than they have been generally found to be, the supply of water in the districts affording a great proportion of the ores of this description, is exceedingly limited, and that the inconvenience and expense attending the dilution of the argentiferous mineral by a large quantity of silicious and earthy matter, is less than the cost and trouble that would be entailed by their concentration.

Patio Process.—The method of extracting silver from its ores, so long employed in the mines of South America, and known as the *patio process*, was discovered in 1557 by Bartolomé Medina, a native of the town of Pachuca, in the neighbourhood of Real del Monte. It is difficult to understand by what course of reasoning a man totally unacquainted with chemical science could have been led to the discovery of a process, of which the *modus operandi* is, even now, to a certain extent, a disputed question, and of which the efficiency does not admit of being at once tested by means of a simple experiment; but which, on the contrary, requires weeks, and, under certain circumstances, even months, for its completion. Although, however, this process requires a considerable period for the full development of its results, the operation of reduction commences almost immediately, and we can therefore only suppose that Medina, being aware of the affinity of mercury for silver, and having mixed this substance with silver ore, sulphate of copper, and common salt, found that a certain portion of the silver had entered into combination with mercury. By keeping this mixture for some time, and occasionally testing the amount of silver taken up by the mercury, which could be readily ascertained by taking a weighed portion of the amalgam and driving off the quicksilver by heat, it would be found, that, for a certain period, the proportion of silver gradually increased, but subsequently remained without change. It is therefore probable that some simple series of experiments of this kind may have conducted him to the discovery of a process which has been of such vast importance, not only to Mexico, but to the world at large, and has so materially affected the total production of silver. A Peruvian, of the name of Carlos Corso de Leca, discovered, in 1586, the method of reduction by iron, *el beneficio de hierro,* which consists in adding to the torta small pieces of iron, by which the reduction of chloride of silver is effected, attended by a corresponding saving of mercury. This process does not, however, appear to have been ever extensively employed.

The next modification in the process of amalgamation was introduced in 1590 by Alonzo Barba, which consisted in conducting the operation in large copper vessels heated by means of a fire placed beneath them. This process, called *el beneficio de cazo y cocimiento,* effected a considerable saving of mercury, attended by a corresponding loss of copper, since the chloride of silver is reduced at the expense of the vessel in which the amalgamation is carried on. This process, by which the reduction of the chlorides is readily effected, but which is not equally applicable to sulphides, was introduced into Europe in 1784 by Baron de Born, an Austrian mining officer, by whom its use in the Hungarian mines was recommended, and from which ultimately sprung, in 1790, the barrel process of Freiberg. It would, however, appear from a work on Metallurgy by Schlüter, printed in 1738, that a process of amalgamation in barrels, or rather vertical tubs, was in use at Kongsberg, previous to that period.

Not having had an extensive practical experience in the extraction of silver from its ores by this method, we have availed ourselves of the descriptions given of it by Humboldt, Duport, and other competent authorities, as well as of data contained in various foreign journals, and in a paper by Mr. J. Napier, jun.* We have also had access to notes made by various friends long engaged in different large reduction establishments in Mexico, and who have furnished valuable statistics relative to the cost of the different operations.

The materials necessary for the reduction of the ores of silver by the patio process, are *magistral*, common salt, and mercury; but in addition to these, sulphate of copper, precipitated copper, and copper and zinc amalgams are occasionally employed.

Magistral.—This is manufactured from copper pyrites, or raw magistral, of which mines occur in many parts of Mexico, but particularly in the district of Tepezala, about twenty leagues south-east of Zacatecas, and sixty from Guanaxuato, from which a large supply is obtained.

The following estimations of copper, made by Napier, show the average percentage of that metal contained in ores from this district:—

$$\begin{array}{l}\text{Copper per cent.}\\ \left.\begin{array}{l}\text{No. } 1=13{\cdot}00\\ \text{,, } \ 2=\ 7{\cdot}47\\ \text{,, } \ 3=13{\cdot}75\\ \text{,, } \ 4=\ 9{\cdot}00\\ \text{,, } \ 5=12{\cdot}50\\ \text{,, } \ 6=10{\cdot}50\\ \text{,, } \ 7=\ 8{\cdot}60\\ \text{,, } \ 8=\ 9{\cdot}40\\ \text{,, } \ 9=\ 8{\cdot}73\end{array}\right\} =10{\cdot}32 \text{ mean percentage of copper.}\end{array}$$

The copper ore, when brought to the works, is first reduced to a coarse sand by dry stamping, and then ground to a fine powder in arrastres. The ground ore is removed from the arrastre to an enclosure, where the water, with which it has been mixed during the process of grinding, is allowed to evaporate; it is then left exposed for a long time to atmospheric influences, as it is generally believed to afford a larger proportion of sulphate of copper by roasting, if previously exposed for some months to the action of the air. The furnaces in which the calcination is effected are called *comalillos*, and have a double hearth, of which the roof is almost flat, with a fireplace at the side.

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* "On the Mexican Method of Amalgamation," by James Napier, Jun., late Chemist and Assayer to the Guanaxuato Mint, Mexico. *Mining and Smelting Magazine*, vol. i. p. 101.

About eight arrobas, or two hundred pounds, of ground ore, with which a few handfuls of salt have been previously mixed, are charged on each hearth. The heat is then gradually raised, and the ore kept constantly stirred during from six to eight hours, when the doors are closed, and the furnace allowed to cool. When sufficiently cold, the doors are again opened, and the charge raked through holes in the bottom of the furnace into arched recesses beneath, prepared for its reception. The percentage of sulphate of copper formed, from an ore of given tenure in copper, depends, to a great extent, on the skill of the workman, and the care bestowed on the operation.

The following table gives the results obtained from the roasting of different parcels of copper ores at the various reduction works at Guanaxuato, as determined by Napier:—

Name of Hacienda.	Sulphates per cent.		Total percentage of Sulphates.	Copper not in the state of Sulphate.		Percentage of Copper in Ore before calcining.
	Copper.	Iron.		As Sulphide.	As Oxide.	
San Joaquin .	40·99	9·73	50·72	4·50	2·11	15·30
„ Nicolas .	20·50	12·38	32·88	2·50	0·23	7·83
Granaditas . .	34·37	6·95	41·32	3·78	2·47	14·00
Salgado . . .	24·64	7·40	32·04	...	2·50	8·00
Pardo . . .	33·18	6·75	39·93	3·00	3·50	12·80
Pastita . . .	31·62	9·05	40·67	0·70	0·15	8·75

In the above table the amount of sulphate of iron is given, as well as the proportion of sulphate of copper, since it has been found that sulphate of iron, as well as the former salt, may be considered an agent in the reduction of the ore, although its action is neither so rapid nor so complete. A small portion of the copper estimated as sulphate will also be in the state of chloride, produced by the chemical changes resulting from the decomposition of the common salt added to the charge. When the ores treated contain either oxide or carbonate of copper, it is usual to add to them a certain amount of iron pyrites, which, by supplying sulphur, assists in their conversion into sulphates. The sulphate thus obtained, being in an anhydrous state, becomes heated on the absorption of water, and this circumstance is taken advantage of for the purpose of making a rough estimate of the quality of prepared magistral, and determining the proportion it will be necessary to employ.

For this purpose the amalgamator takes a small quantity of the calcined ore in his hand, and then gradually inserts it into water, judging from the degree of heat developed, of the quantity of sulphates contained in the parcel. It is also considered by some amalgamators advantageous to employ the magistral as soon after its calcination as possible, since by absorbing moisture its action is said to become less energetic, and a larger amount to be required. The quantity of water, however, which could be absorbed from the atmosphere by the magistral, is so small, in comparison with that contained in the argentiferous mud with which it is incorporated, that the moment they become mixed the whole of the anhydrous sulphate present must at once become fully hydrated; and it is therefore difficult to ascribe the belief that it is necessary to employ freshly calcined magistral, to anything but prejudice on the part of those who entertain the opinion. That such is in reality the case, appears to be evident from the fact that the whole of the sulphate of copper derived from the operation of parting at the various mints, is purchased for the purpose of being employed in lieu of magistral, and that preference is now given to this hydrated salt by the more intelligent amalgamators.

Salt.—In addition to the salt imported from the sea-coast, a very large proportion of that employed in the mining districts of Mexico is obtained from various lagunes situated at more or less considerable distance from the mines. Large quantities of impure salt derived from this source, and forming an efflorescence on the surface of the ground during the dry season, were formerly employed under the name of *saltierra* for the purpose of amalgamation. The cost of transporting large quantities of such an impure material, added to the great increase, in bulk, of the tortas, caused by adding the amount necessary to furnish the required quantity of chloride of sodium, has, however, caused its use to be nearly abandoned; the salt is therefore now generally concentrated by lixiviation and evaporation before being brought to the mines.

The principal localities from which saltierra was formerly derived, and where salt is now manufactured for mining purposes, are Salinas, and Peñon-blanco, there being in both localities, in addition to the saline incrustations before referred to, springs holding chloride of sodium in solution. The following analysis, made by Berthier, of the saltierra from Peñon-blanco, gives the composition of the material formerly employed instead of salt, and shows how large a proportion of its constituents must have been totally inert when introduced into the torta:—

Chloride of Sodium	19·00	soluble in water.
Sulphate of Soda	2·20	
Carbonate of Lime	13·60	insoluble in water.
" Magnesia	1·60	
Oxide of Iron	9·80	
Clay and Sand	46·20	
Water and Organic Matter	7·60	
	100·00	

The amount of chloride of sodium contained in the purified salt from Peñon-blanco is usually from 80 to 85 per cent.; the remainder, for the most part, consisting of sulphate of soda.

The following table gives the composition of four samples of salt supplied by the district of Salinas, analysed by Napier:—

	No. 1.	No. 2.	No. 3.	No. 4.
Chloride of Sodium	96·623	91·141	90·422	86·853
" Magnesium	0·008	2·538	2·520	0·044
" Calcium	0·114	1·574	1·310	0·125
Sulphate of Potash	Trace	3·141	3·556	0·029
" Soda	3·255	...	...	12·949
" Lime	...	1·606	2·192	...
	100·000	100·000	100·000	100·000

The purest description of salt employed in Mexico is that brought from the sea-coast, which often contains as much as 95·50 per cent. of chloride of sodium, but its great cost prevents its general employment in the reduction works of the country.

Mercury.—Nearly the whole of this metal, of which the consumption in the Mexican mines is very large, is imported from Europe and California, although it has been found in various localities in the empire, particularly at Mazapil and the Gigante, near Guanaxuato. These mines are not, however, now worked, although considerable quantities of quicksilver have, at various times, been obtained from them. The district of El Doctor, 150 miles north of the city of Mexico, has also furnished a certain amount of this metal.

Lime.—This substance cannot be regarded as an essential in the patio process of amalgamation, since it is only employed when too large a proportion of magistral has been added to the torta, and is then used for the purpose of decomposing the excess of sulphates. Wood ashes are also sometimes made use of for the same purpose.

Copper Precipitate.—Instead of using lime for the purpose of counteracting the effects of any excess of sulphate of copper which may have been added, precipitated copper is employed in many of the haciendas, and acts by the reduction of the metallic chlorides. The use of copper precipitate in the process of amalgamation by the patio was first introduced by Mr. Louckner, who also, in conjunction with Mr. Mackintosh, obtained a patent for the employment of copper amalgam in the torta, with a view to reducing the loss of mercury. This patent was, however, readily evaded by the use of precipitated copper, which, by combining with the mercury, produced the same effect.

Composition of Ores.—The ores subjected to patio amalgamation differ somewhat in their composition in the various mining districts. The following analysis gives the composition of an ore of more than average richness, from the district of La Luz, Guanaxuato :—

Silver	1·04
Iron	4·71
Copper	0·55
Sulphur	6·79
Carbonate of Lime	8·25
„ Magnesia	3·26
Silica	75·00
	99·60

The ores of Real del Monte vary considerably in their composition, but often contain manganese, antimony, and lead; and are consequently difficult of reduction by the patio process of amalgamation.

A sample of ore from the Santa Brigida vein, analysed by Mr. Rogers, afforded him the following results :—

Silver	0·25
Sulphide of Copper	0·40
„ Zinc	2·30
„ Lead	2·82
Peroxide and Sulphide of Iron	7·50
„ of Manganese	5·30
Carried forward	18·57

Brought forward	18·57
Alumina	8·00
Carbonate of Lime	1·45
" Magnesia	1·60
Silica	68·00
Antimony, Tellurium, traces of Gold, Potash, and Soda	2·38
	100·00

An average sample of the ores worked in the district afforded the following result:—

Sulphide of Silver	0·15
" Iron	26·52
" Lead	2·07
" Arsenic	0·10
" Zinc	5·60
Sulphate of Iron	0·25
" Lime	0·43
Peroxide of Manganese	3·54
Carbonate of Lime	4·18
" Magnesia	0·96
Silica	50·00
Moisture	6·80
	100·00

In the mining districts of Mexico, as well as in Nevada and other localities producing silver ores proper, there are immense accumulations of veinstone containing in the aggregate vast quantities of silver, but of which the yield is not sufficiently high to admit of their being treated with profitable results. Numerous plans have been proposed, and various machines invented, with a view to the concentration of these ores by washing, but, as before stated, none of them have hitherto been found capable of extensive practical application.

Rough Stamping.—The ores to be subjected to the process of patio amalgamation are usually first crushed dry, to the state of coarse gravel in a stamping mill, and subsequently reduced by porphyrisation in the arrastre to the necessary degree of fine division. The former machines generally consist, in Mexico, of eight wooden lifters or stems, shod at their lower extremities with iron, and set in motion by cams, worked by a water-wheel, or more frequently by a team of mules. The ore, as fed to these *morteros* or stamping mills, is in pieces of about the size of the fist; and, as it is broken, falls into sieves formed of hides perforated with small round holes, and fixed in an inclined position before the battery: the portion of the ore which passes through these sieves is removed to the arrastres to be further ground, whilst the coarser particles, remaining on the skins, are again thrown back into the mill. In the majority of the reduction-establishments of Mexico this work is entirely performed in mills worked by relays of mules which are driven at a rapid rate, and frequently replaced by a fresh team. Each *molino* requires eighteen mules, which are successively harnessed to the machine in sets of three. An apparatus of this description will, in the course of twenty-four hours, reduce about five tons of ore to a state suitable for treatment in the arrastre.

Fine Grinding.—The arrastre or *tahona* employed for this purpose is constructed as described, page 168: but is paved with great care and is very strongly put together. The bottoms are constructed of hard porphyritic stones about thirty inches in length, placed vertically, and the interstices carefully filled with *cabecilla*, which is the residue remaining after the washing up of a torta. The sides are formed of either wood or flagstones, which stand about two feet above the level of the floor. Their diameter in the haciendas of Guanaxuato is 4½ varas, or 12 feet 4 inches, arrastres of this size being called *arrastres de marca.* The *voladoras*, or mullers, are of porphyry, and have a length somewhat less than the radius of the arrastre, with a thickness of about sixteen inches. In each of them are bored two holes, into which are driven wooden pegs for attaching the chains or straps of raw hide by which they are connected with the arms traversing the vertical shaft. Each arrastre is provided with four voladoras, and is worked by a team of two mules.

In certain districts, as at Real del Monte, basalt is sometimes employed for mullers, but, although sufficiently hard for this purpose, it has a very close grain, and consequently, when once it has become smooth, it passes over the ore without causing the grinding effect produced by porphyry, which being coarser in its texture, retains a good rubbing surface to the last. As soon as the bottom of an arrastre has been put in, a new voladora is set to work to grind cabecilla, to which a certain quantity of water has been added, and by which the interstices between the stones are gradually filled. This is continued with one muller during the first day, and on the second another is added, with which the operation is carried on until the end of the third day, when another grinding stone is attached, and the treatment of poor ores may be commenced. After the expiration of four or five days, the fourth stone is put on, and the surface of the grinders and of the bottom of the arrastres, having now attained their normal condition, ores of the usual degree of richness may be charged into the arrastre, and fine grinding carried on in the usual way.

Extraction of Gold.—In districts such as Guanaxuato, where the ores contain, in addition to silver, minute quantities of gold, the latter metal could often not be advantageously separated from the former if obtained together in the form of an alloy; but by keeping the arrastre constantly charged with a certain quantity of mercury, or with an amalgam of silver or copper, the gold is so concentrated in the resulting amalgam, as to afford highly profitable results. This method of effecting the concentration and extraction of gold appears to have been long employed in some of the mining districts of Mexico, but Humboldt states, that at the time of his visit to Guanaxuato in 1804, it had not been introduced into that neighbourhood.

At Guanaxuato, where the ore is more finely ground than in any other district in North America, an arrastre is charged with from six to eleven quintals of *granza* or coarse sand from the stamping mill, to which are added about ten gallons of water, which is sufficient to bring it to the state of a thick mud; and on the amount of water thus added, will very much depend the degree of fineness to which the ores can be reduced. If the arrastre be a new one, or one from which the amalgam has previously been removed, from five to ten pounds of silver amalgam are introduced. For this purpose some amalgamators prefer a very dry amalgam, whilst others employ a larger proportion of mercury; but it appears to be generally admitted that a tolerably dry amalgam is to be preferred, since, if it contained a large quantity of mercury it would be liable to run into the crevices of the bottom, and thus cease to be of any advantage to the operation. When, on the contrary, the amalgam made use of is moderately dry, it not only spreads itself over the surface of the bottom, under the influence of the grinding action of the voladoras, but is also believed to possess a stronger affinity for gold, and to form with it an amalgam more readily than pure mercury would under similar circumstances.

The arrastres used in the haciendas of Guanaxuato are in most cases charged each morning at four o'clock, when, as before stated, a barrel of water, containing about ten gallons, is introduced at the same time with the ore; at nine o'clock another barrel, or sometimes a barrel and half is poured in, and at twelve o'clock one barrel, at three o'clock three barrels, and at four, five barrels are added. The quantity of water used, will however somewhat depend on the nature of the ores operated on. As the operation of grinding progresses, the amalgam by degrees accumulates in the crevices in the bottom of the arrastre, and every alternate morning a small sample or *tentadura* is taken from the amalgam in the bottom of each arrastre, cleaned well by washing, and its condition carefully examined. Its dryness or moistness is judged of in accordance with its behaviour when pressed by the thumb against the side of the jicara containing it, and from the result thus obtained is estimated the amount of mercury, if any, that is required to be added to the arrastre previous to the next grinding. The usual amount of mercury added is about half a pound every second morning, although this will in a great measure depend on the richness of the ores that are being worked, but as a general rule the amalgam should not contain above twenty per cent. of gold and silver. The amalgam is usually removed from the arrastres every three months, but in some instances they are cleaned up at even longer intervals. At the expiration of twenty-four hours, when the grinding is completed, the *lama*, or slime, is baled out into a barrel, in which it is removed to reservoirs, formed in masonry, from which a portion of the water becomes evaporated by exposure to the sun and air, and leaves the mass in a fit condition for subsequent treatment in the patio. In some establishments, instead of the lama being transported in barrels to these reservoirs, it is baled into launders by means of which it is conducted to the proper receptacles; whilst in others it is tapped from a plug hole at the bottom of each arrastre directly into these launders. When the lama is dipped out in a batea, a plate of iron, or some other metal, is laid on the bottom of the arrastre for the purpose of preventing any of the rich amalgam from being scraped up by the batea and becoming mixed with the slimes; after this operation, which occupies at most about half an hour, is completed, the arrastre is charged with a fresh quantity of granza and the process of grinding is again proceeded with, as before described. The mules, by which the machine is worked, are changed every six hours, the arrastres being generally arranged in a double row on either side of a long shed or *galera*.

The augmentation of weight which takes place during the operation of grinding, and due to the wearing away of the voladoras and bottom of the arrastre, usually amounts to from eight to ten per cent. The bottom will often continue in good repair during twelve months, but the grinding stones or mullers seldom last beyond a month, and are sometimes worn out considerably within that time.

Raspar.—The removal of the amalgam from the arrastres is called the *raspando* or scraping, and takes place more or less often in accordance with the quality of the ores under treatment. The arrastres are, however, seldom cleared of amalgam more often than once in three months, and in some instances only once in six months. This is done by removing, with a hooked iron scraper, all the substances which have accumulated between the stones forming the bottom of the arrangement, and which consist of a mixture of coarse fragments of gravel, finely ground ore, and an amalgam of silver and gold. If the bottom of the arrastre has been entirely worn out it is removed, and every stone well scraped and cleaned, after which a new bottom is put in as before described.

The substance collected either by scraping out the interstices between the paving of the arrastre, or by cleaning every portion of stone forming part of the old bottom, is next carefully washed in bateas in a tank filled with water. For this purpose, a portion of the stuff to be washed is placed in the wooden bowl, and a little fresh mercury added to it, for the purpose of collecting the finer particles of amalgam, and preventing them, as much as possible, from being floated off on the water. In this way the principal portion of the amalgam is collected in the bowl, whilst some of the smaller particles, mixed with the lighter earthy impurities, become deposited in the bottom of the tank, from which they are subsequently removed for the purpose of being re-washed on an inclined table, called a *planilla*. The amalgam, after being collected and cleaned, is first freed from a portion of its mercury by straining through a leathern or canvas bag, and then retorted and sent to the Government Assay Office, where it is assayed and melted into bars, and then forwarded to the Mint, for the purpose of being converted into coin.

Loss of Gold.—The yield of gold obtained by this method of treatment is considerably less than the total amount contained in the ores, as indicated by assay; the loss experienced being generally believed to vary from 25 to 40 per cent. on the assay produce. Various circumstances contribute to produce this deficit in the gold. In the first place, there is always a loss of finely-divided amalgam during the process of washing up the various substances obtained by scraping the bottoms of the arrastres. Secondly, the silver subsequently extracted by amalgamation in the patio, invariably contains a small proportion of gold; and thirdly, the *polvillos* remaining from the washings of the *tortas*, and which principally consist of argentiferous iron pyrites, also retain a certain amount of gold. These polvillos, or *relaves*, are subjected to a process of roasting in a reverberatory furnace, and a second grinding in the arrastre, after which they are a second time subjected to the process of patio amalgamation.

Loss of Mercury in the Arrastre.—It was formerly believed that by the process of grinding the ores of silver and gold with mercury, those portions only of the metals which exist in the metallic state are obtained in the form of amalgam; but this is evidently not the case, since there can be no doubt that more or less of the sulphide of silver is always reduced to the metallic state at the expense of a corresponding amount of mercury, which is itself converted into sulphide. It is also generally well known to *azogueros*, or amalgamators, that to obtain a given weight of the precious metal by this process, a nearly equivalent quantity of mercury must be sacrificed, and this could by no means be accounted for, on the supposition of its uniting only with the free gold and native silver present in the ores.

The following figures are given by Napier as the result of an experiment made at one of the reduction works of Guanaxuato, and go to show that the loss of mercury is mainly due to its combination with sulphur, derived from the reduction of sulphide of silver:—

The amalgam added to an arrastre weighed 70 lbs., and was composed of 14 lbs. of silver combined with 56 lbs. of mercury.

The mercury added, exclusive of that in the amalgam, weighed	330 lbs.
Mercury in amalgam	56 „
Plata Mixta, gold and silver obtained, including silver added in form of amalgam	84 „
Weight of amalgam which should have been obtained	470 lbs.

The amalgam actually produced only weighed, however, 400 lbs., and consisted of 84 lbs. of gold and silver, and 316 lbs. of mercury, showing a loss of 70 lbs. on the total quantity of quicksilver introduced into the arrastre. The alloy obtained, consisted of 18 lbs. of gold, and 66 lbs. of silver; and as the gold, being in the metallic state, must have united directly with the quicksilver without entailing any loss of that metal, the deficit may be regarded as equal to the weight of the silver taken up, the slight excess of loss observed, being readily accounted for by mechanical loss in washing, and the escape of mercurial vapours during the subsequent process of retorting the amalgam. The ores of Guanaxuato contain scarcely any native silver, gold being practically the only metal present in the metallic state, and consequently the loss of mercury can only be ascribed to the reduction of sulphide of silver by mercury, and its gradual conversion into the sulphide of that metal.* When copper amalgam, instead of an amalgam of silver, is introduced into the arrastre, the copper gradually disappears, and is replaced by silver, the loss of mercury being at the same time less than under ordinary circumstances. When the ores to be treated do not contain a sufficient amount of gold to render its extraction a profitable operation, they are ground in the arrastre without the addition of mercury; and in some localities, as in Zacatecas, the grinding is not so long continued, and the ore is consequently reduced to a less finely divided state. In the locality last named, an arrastre of similar dimension to those employed at Guanaxuato, grinds ten quintals in the course of thirteen hours, but the resulting lama is much coarser, and it is probable that the less satisfactory results generally obtained in this district may, in a great measure, be attributed to that circumstance.

The Patio.—The patio is a large court-yard, generally paved with flagstones, of which the joints are carefully cemented in order to prevent the loss of mercury which would otherwise take place. This flooring has given to it a slight inclination, in order that any water falling on it may the more readily run off. In some cases, however, as at the hacienda of Regla, near the Real del Monte, a wooden flooring is employed instead of a stone one; and the patio, comprising an area an acre and a half in extent, is carefully covered with a wooden floor, on which a thousand tons of argentiferous slimes, mixed with thirty tons of salt, three tons of sulphate of copper, and eighteen thousand pounds of mercury, are constantly spread in different stages of the process of amalgamation. Wooden floors were also employed in the various reduction works near Virginia City, into which the patio process was introduced, but as the climate of Nevada was not favourable to this system of amalgamation, its use has, for the present, been discontinued. The following woodcut, Fig. 40, represents the patio at the Gould and Curry Works, near Virginia.

The ground slimes, on their removal from the arrastres or tahonas, are deposited, in an almost liquid state, in walled receivers, called *cajetes* or *lameros*, where a portion of the water is removed by evaporation, and where it is allowed to accumulate until there is a sufficient quantity to form a heap or torta, which at Guanaxuato usually consists of sixty montones.* When the amount of lama necessary for a torta has been collected in the cajete, it is carried out into an enclosure formed on the patio, about thirty feet in diameter, generally made by laying on each other square beams of wood, kept in their places by large stones, and made tight by filling the joints either with clay or horse dung. Into this the lama is introduced, until it forms a layer of about a foot in thickness, and is allowed to remain until by

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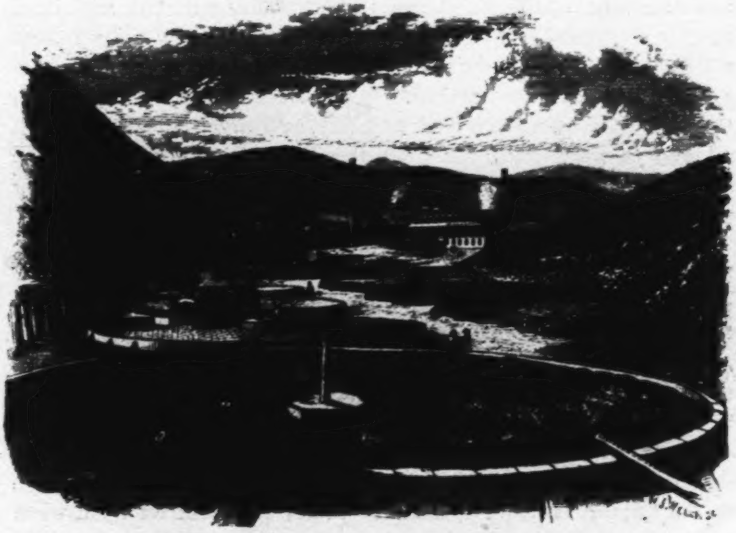
* The equivalent of silver being 108, and that of mercury 101, sufficiently explains why the loss of mercury should about equal the weight of silver taken up by that metal.

FIG. 40.



PATIO AT THE GOULD AND CURRY, NEVADA.

(*From a Photograph.*)

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* The weight of the monton varies in different localities—

In Guanaxuato a monton usually contains	32	quintals
„ Real del Monte, Pachuca, and Tasco	30	„
„ Zacatecas and Sombrerete	20	„
„ Fresnillo	18	„
„ Bolaños	15	„

the evaporation of the water, it has gained the consistency of a rather thin mud. This condition being arrived at, the amalgamator proceeds to *ensalmorar*, *i.e.* from three to five per cent. of salt is added in accordance with its quality, and the nature of the ores under treatment. It is, however, well known, that the larger, up to a certain point, the proportion of salt employed, the more rapid will be the action of the torta, although many amalgamators never employ above three per cent. of this substance, notwithstanding that an additional two per cent. would have the effect of completing the operation at least six days earlier; but the time thus gained is not found to compensate for the extra expense entailed. When the salt has been added to the torta, it receives the first treading, or *repaso*, after which it is allowed to stand until the following day, when the whole of the salt will be found in a state of solution, and thoroughly and regularly mixed with the lama composing the heap.

Incorporar, &c.—The day after the salt has been thus mixed with the lama, the addition of magistral and mercury takes place. For this purpose the torta is, if necessary, brought to the proper consistency by the addition of water, and the magistral thrown evenly over its surface by means of wooden shovels. The proportion of this reagent to be added varies, to a certain extent, in accordance with its richness in sulphate of copper; but in the case of employing magistral of the usual strength, something less than one per cent. is generally found sufficient. As soon as the magistral has been spread over the surface of the torta, it is again trodden by mules for about an hour, when the mercury necessary for the completion of the operation is generally added, the quantity required being from 3½ to 4 lbs. for every mark of silver supposed to be contained in the heap. The introduction of mercury is effected by making it run through a linen cloth in such a way that its particles may be divided in the state of minute globules, and great care must be taken that the torta be not too wet, since in that case the mercury will be liable to collect in large masses. The lama should not, on the other hand, be too dry, as the mercury would then become very finely divided, and a considerable loss, both of that metal and silver, takes place in the washing up of the torta. The exact consistency to be given to the metalliferous mud, in order to obtain the best possible result, can only be determined by practice; but it should be in such a condition that the feet of the animals treading it may pass readily through it, and yet leave behind them distinct prints of their hoofs. After the addition of the mercury the torta is again trodden for about four hours, in order to effect its intimate mixture throughout the whole mass. When crystallised sulphate of copper is employed in lieu of magistral, from 7 to 9 lbs. are added for each ton of ore contained in the torta.

When the magistral and mercury have been added to the torta, and it has received the first treading, chemical action at once commences, and it becomes necessary that the azoguero should closely watch its operation, by taking frequent samples or tentaduras; the colour and general appearance of the mercury being the chief guides by which he is enabled to judge of the progress of the operation. Shortly after the *incorporo*, the azoguero takes a sample collected from thirty to forty different places, in the torta, and washes it carefully either in a jicara or in a horn spoon, similar to that sometimes employed for testing samples of gold quartz in California, and, after removing the earthy particles, carefully examines what remains, which consists of *polvillos*, or metallic sulphides and mercury. The latter at this stage of the operation contains but little silver, and its colour and state of division afford the only indications of the more or less satisfactory working of the torta. If the mercury be found divided into minute globules, or if its natural colour has been but little changed, except that it has acquired a slightly yellow tinge, it is considered a proof that a sufficient amount of magistral has not been added. When, on the contrary, the mercury is of a deep grey, or leaden hue, it is evident that there has been too large a proportion of magistral introduced into the torta, which is said to be *hot*, and it may be found necessary to add a little lime in order to prevent an undue loss of mercury. When the heap is in good working order, the surface of the mercury presents a distinct light grey appearance, and although it is of course desirable to at once introduce such a proportion of magistral as will immediately throw the torta into good working condition, it is better that the amount of this ingredient should be too small, than that it should have been added in excess. When the magistral has been added in proper quantity the appearance of the tentadura will, on the day after the repaso, be found to have considerably changed. The mercury, if now pressed with the thumb against the side of the jicara, or bowl, is found to contain silver amalgam, and what, at the first trial, appeared as *desecho*, or broken-up mercury, has now become *limadura de plata* of a whitish colour and in the form of thin scales, which, when rubbed, are found to consist of dry silver amalgam, or *pasilla*. When this makes its appearance shortly after the addition of magistral and mercury to the torta, it is a proof that the former has been added in proper proportion, and that the operation is progressing favourably.

To make a tentadura, a fair sample of about eight ounces, taken from different parts of the torta, is placed in a jicara, and slowly washed so as to remove the lighter portions only, leaving in the bottom the limadura de plata, the mercury with its *lista* or tail, and a portion of the heavy sulphide of iron. A little water is now taken in the jicara, and a peculiar motion given to it so as to arrange its contents in the following order: the limadura occupies the upper portion of the bowl, next come the sulphides contained in the ores, and last of all the mercury and amalgam, in the form of a large globule. As the most important indications are to be obtained from the condition of the limadura, this is first examined by holding the bowl in an inclined position in the right hand, whilst with the thumb of the left hand it is rubbed against the side of the vessel. Whilst this is being done, its colour is carefully observed, as well as the facility with which it can be converted into amalgam, and the consistency of the amalgam so produced. The metalliferous portions of the ore are generally passed over as not affording any information relative to the progress of amalgamation, but the globule of mercury at the bottom is examined with regard to its colour and the amount of amalgam it contains. The latter is determined by pressing it with the thumb against the side of the bowl. Three tentaduras are usually made on each torta daily; one in the morning before commencing to tread, another after it has been trodden for some time, and a third after the repaso has been completed. In selecting samples for the purpose of making a tentadura, it is necessary to take portions, not only from the surface of the heaps, but also from the interior, since the top from being exposed to the action of the sun and air is always in a more advanced condition than the middle of the heap.

The treading of a torta has the effect of stimulating the action of the magistral, and is repeated, every alternate day, as often as the tentaduras indicate a necessity for doing so. Formerly, the mercury was not all introduced at once, and after from fifteen to thirty days, according to the season, &c., it was found to be no longer in a liquid state, but had been wholly transformed into a dry amalgam from which quicksilver could no longer be obtained by pressure between the finger and thumb. A further addition of mercury, amounting to three-twelfths of the total quantity, was now made, which in the course of about ten days was, in its turn, converted into dry amalgam, when the last twelfth was added; and this was not, in most cases, entirely converted into solid amalgam. If, however, this took place, more mercury was added, until the further action of the torta had no longer the effect of converting it into solid amalgam, when the heap was said to be *rendido*, or to have yielded the whole of the silver which could be obtained from it. It is, however, now usual to add all the mercury immediately after the introduction of magistral.

If in the course of the operation it be observed that after repeating the repasos no augmentation takes place in the quantity of limadura and solid amalgam, the torta is said to be *cold*, and an additional amount of magistral is introduced, the process of amalgamation being subsequently continued until the desired effect has been produced. When, on the contrary, the mercury is covered by a dark grey coating, and the limadura is replaced by desecho and finely-divided mercury, the torta is said to be too "hot," and an addition is made of a little lime or wood ash, the treading being also suspended until it has been ascertained, by repeated tentaduras, that the torta has again recovered its normal condition. The azoguero never has, however, recourse to the introduction of lime or ashes except in extreme cases, since these reagents have the effect of retarding the operation, and decreasing the yield of silver, without effecting the revivification of the mercury already uselessly expended.

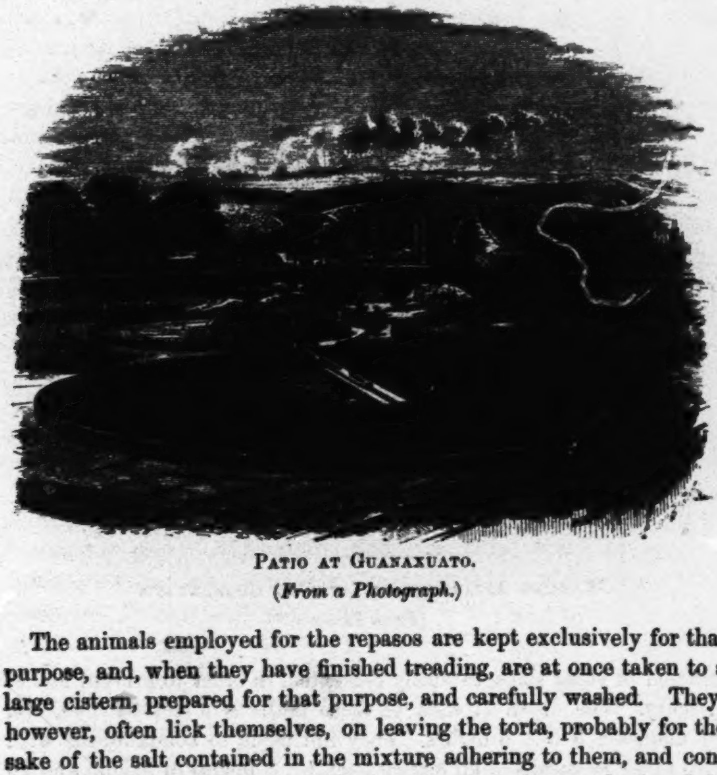
The amalgamators of Guanaxuato, when they are well acquainted with the nature of the ores under treatment, do not hesitate to work their tortas rather hot, and consider that by doing so they not only save time, but also obtain a larger yield of silver, without incurring a greater loss of mercury. At Zacatecas and Fresnillo, where the ores contain considerable quantities of mixed sulphides, it is customary to employ a larger proportion of magistral, particularly when a notable amount of galena is present. From this circumstance the mercury frequently presents an appearance which would cause much uneasiness to an azoguero of Guanaxuato, but when the torta shows symptoms of becoming too much heated, it is allowed to remain some days without treading, and after a time again falls into good working order without the addition of either lime or ashes. When the action of the torta has almost ceased, and nearly the whole of the silver which the process is capable of extracting from the ore, has been taken up by the mercury, the limadura becomes weak, and on being rubbed with the thumb shows but little solid amalgam. As soon as, on examination, it is found to be free from amalgam, and runs together in globules in the bottom of the jicara, the operation is generally considered finished, and the torta is said to be "rendido."

In some cases, however, the limadura may show signs of a torta being finished, when in reality such is not the case; and for this reason the ground sulphides or polvillos must also be examined by extending them over the interior surface of the jicara, and rubbing with the finger the small metallic globules with which they are mixed. If these readily unite into a large globule of running mercury, there can be no doubt that the torta is rendido; but if, on the contrary, they yield dry amalgam, such is evidently not the case, and the operation is further continued.

The amalgamators of the present day, however, do not entirely depend for their guidance on results obtained by washing, but also take an average sample from the torta, which being assayed, the total amount of silver contained in it can be readily calculated therefrom. Another weighed quantity of the same sample is carefully washed and the mercury and amalgam collected, which, on being assayed, afford a result from which may be calculated the total amount of silver which has been removed by amalgamation. In most instances, also, the mercury, instead of being added to the torta at different periods, is now all introduced immediately after the addition of the magistral.

Repasar.—The treading of the tortas is effected by means of mules or horses, the former being most frequently employed, and is repeated every alternate day until the operation is completed. The mules used for this purpose are tied together four abreast and blindfolded, one mule for every two montones of ore being required for the effectual treading of a heap. The operation is directed by a driver, who stands in the centre of the torta and guides the animals by means of a long halter, causing them to first tread at the edge, and gradually diminishes the radius of the circle described. In this way the circle becomes slowly diminished, and the thorough treading of the whole mass is thereby insured. This work, which commences at six in the morning, is generally completed about one or two o'clock in the afternoon; and in addition to the treading, each torta is turned over twice a week by means of wooden shovels. This operation is commenced immediately after the completion of the day's treading. In some instances, instead of entirely relying for the incorporation of the mass on the action of the feet of the animals, they are made to drag behind them a framework on wheels, which acts in the same way as the ordinary mortar mill. When this is employed it is attached to a long wooden arm revolving on a spindle in the centre of the torta, and in order to allow of the radius being gradually diminished, the arm is provided with slots, in which the central pin readily traverses; thus admitting of the circle described by the wheels being increased or diminished at pleasure. This arrangement is represented, Fig. 41.

FIG. 41.

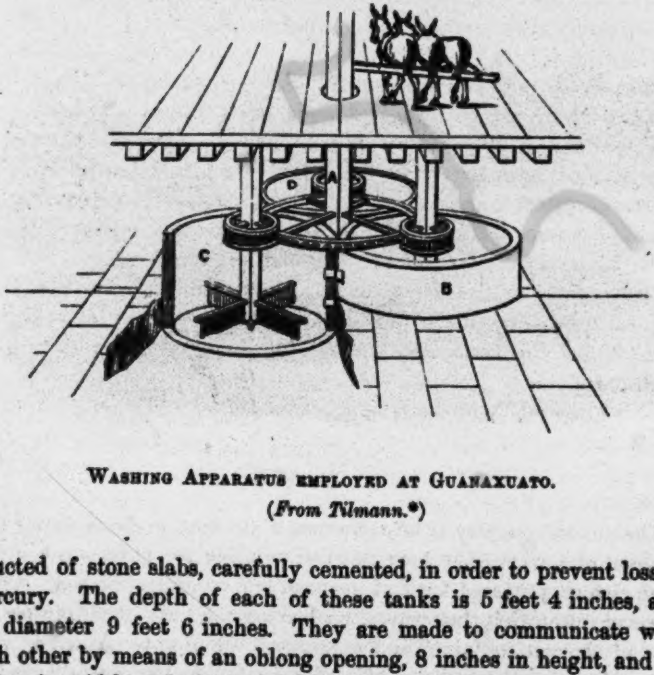


PATIO AT GUANAXUATO.
(From a Photograph.)

The animals employed for the repasos are kept exclusively for that purpose, and, when they have finished treading, are at once taken to a large cistern, prepared for that purpose, and carefully washed. They, however, often lick themselves, on leaving the torta, probably for the sake of the salt contained in the mixture adhering to them, and consequently, when they die, a ball of amalgam is frequently found in their stomachs. These pieces of amalgam sometimes weigh many ounces, and are always exceedingly hard, and contain but a comparatively small proportion of mercury.

Washing.—When the operation of the torta is completed, and the whole of the silver which the process is capable of extracting has entered into combination with mercury, it is rendido, and a portion of fresh mercury, called a *baño,* is in some districts added to the heap; but this is not usually employed in the haciendas of Guanaxuato. The next operation is the separation of amalgam from the various earthy and metalliferous constituents of the torta by a process of washing, which, at Guanaxuato, is conducted as follows:—The *lavadero,* or washing apparatus, consists of three circular tanks, or *tinas* B, C, and D, Fig. 42, built close together in a circle, and constructed of stone slabs, carefully cemented, in order to prevent loss of mercury. The depth of each of these tanks is 5 feet 4 inches, and its diameter 9 feet 6 inches. They are made to communicate with each other by means of an oblong opening, 8 inches in height, and 10 inches in width; of which the first is placed at a height of 8 inches,

Fig. 42.



Washing Apparatus employed at Guanaxuato.
(*From Tilmann.**)

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* From "Bergbau und das Amalgamations-Verfahren in Guanajuato in Mexico, vom Königl. Preuss. Berg-Referendar E. Tilmann," Münster, 1866.

and the other at a distance of 30 inches, from the bottom of the tanks. In addition to these, the last tank is provided with two separate discharge holes; the first at a height of six inches from the bottom, and the other, which is only opened for the purpose of cleaning up, is situated close to the bottom. The diameter of the upper opening is about four inches, and that of the lower one inch.

In the middle of each tank is an upright wooden shaft, carrying four arms, furnished with wooden teeth, acting as agitators; the whole being set in motion by a central shaft A, provided with a spur wheel, working in pinions on the tina shafts; the central shaft, passing through the upper flooring, is turned by means of mules, attached to an arm fixed horizontally for that purpose.

The pinions giving motion to the agitators in the second and third tanks are a little larger than that working the stirrer in the first, and consequently their motion is somewhat slower.

The tank B, into which the metalliferous slimes from the torta are first charged, is called the *tina cargadora;* whilst the last D, from which they are run off after passing through C, is called the *descargadora*, or discharger. Before being taken to the lavadero, the torta is first divided into several parcels, each of which is softened by the addition of water, and subsequent treading, and then carried to the washing house in large bateas, dusted on the inside with dry horse-dung, in order to prevent loss. About three montones of lama are now measured, by means of large wooden bowls, and gradually introduced into the first tank, water being at the same time run in, and the machinery made to revolve rapidly by driving the mules at a gallop.

When the whole of the lama has been introduced, the speed of the machinery is gradually reduced, until the mules, which are specially trained for the work, move only at a gentle walk; whilst the azoguero from time to time dips out a portion of the slime, and washes it in a jicara, for the purpose of ascertaining if it still retains an appreciable amount of mercury, or whether it is sufficiently free from quicksilver and amalgam to safely admit of running off the earthy particles in suspension. When the washings of the samples taken from the tinas afford only minute metallic traces, the plug at some distance above the bottom of the descargadora is removed, for the purpose of discharging the slimes; and as soon as they have been run off, the plug is replaced, and the operation continued until the whole of the torta has been washed up. At Guanaxuato, the weight of the torta usually amounts to about 60 montones, although as many as 80 are occasionally treated at one time.

In addition to the amalgam which remains in the bottom of the tinas, there is also a considerable quantity of the heavier constituents of the ore treated. This residue, or *cabezilla*, contains a large proportion of amalgam, which is obtained by a subsequent washing. For this purpose it is removed in wooden bowls to a tank, called the *pila apuradora*, and thrown into large bowls or bateas. The *batea apuradora* varies from three to five feet in diameter, and floats on the surface of the water contained in the tank. The person using the batea leans over the side of the tank, and, with one hand on each side of the bowl, gives to it the peculiar washing motion, taking up a small quantity of water, which, after circulating round the edges of the vessel, is finally discharged, carrying with it a certain portion of the cabezilla. The deposit of finely-divided mineral remaining with the amalgam in the washing apparatus is, at Guanaxuato, generally about one-eighth of the total weight of the torta; and the relaves, resulting from its treatment by washing in bateas, are subsequently re-ground in arrastres. By this means they are made to yield a certain quantity of amalgam, rich in gold, but are not generally a second time subjected to patio amalgamation.

The amalgam thus obtained is carried to the mercury-house, or *azogueria*, where it is deposited in a large stone trough; and as soon as the whole amount produced by a torta has been collected, a large quantity of pure mercury, together with a little water, is added. The mass is now well stirred by hand, for the purpose of causing the separation of impurities which gradually come to the surface, and which are, from time to time, wiped off by means of a woollen cloth. A small quantity of clean water is added after each removal of the impurities, and the operation repeated until the surface of the amalgam presents a bright uniform appearance; when, after being carefully wiped, it is ready for removal to the *manga*, or strainer.

At Zacatecas and Fresnillo the treatment of the ores in the patio is very similar to the process employed at Guanaxuato; but they are generally much less finely ground, and the washing up of the tortas is not similarly conducted. At Zacatecas, an arrastre of the same dimensions as those used at Guanaxuato is made to grind ten quintals of ore in the course of thirteen hours; and this difference in the fineness of the division of the lama in the two localities would lead to the conclusion that this circumstance may, to a certain extent, account for the less satisfactory results obtained at Zacatecas and Fresnillo. The azogueros of these districts, however, maintain that a more complete grinding of the ores would not sufficiently augment the returns to pay the extra expense incurred.

Instead of transporting, as at Guanaxuato, the lama containing the amalgam, in a nearly dry state, directly from the patio to the lavadero, the azoguero, as soon as he considers the torta to be rendido, adds a quantity of mercury, weighing about eighty per cent. of the amount previously employed for the extraction of silver; and, after giving it a repaso, it is carried to the washing house.

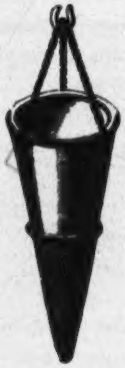
The washing is conducted, in the principal haciendas of Zacatecas and Fresnillo, in a single circular cistern of stone, nine feet in diameter, and seven feet in depth; and of which the bottom is generally made of a single stone, about ten feet in diameter, and a foot in thickness. Two of these tinas are placed side by side, in such a way that one vertical shaft, turned by mules, communicates motion to both; but no other connexion exists between them. The speed of the agitator is greater than that at Guanaxuato, and about two and a half montones of lama are passed through each cistern in the course of an hour. The violent agitation caused by the rapidity of the motion given to the arms, together with the great speed with which the slimes are passed through the apparatus, causes a considerable amount of amalgam to be carried off during the operation; in order to recover which, it becomes necessary to re-wash, by means of the planilla, the heavy residues collected outside the discharge orifice of the tina. The amalgam remaining in the bottom of the circular tank is deposited in an almost pure state, and does not generally require concentration by hand washing. The residues chiefly consist of iron pyrites and other sulphides, forming metalliferous sands, containing more or less silver, according to the nature and richness of the ores originally treated. These, when sufficiently rich, after being previously roasted in a reverberatory furnace, and again ground in an arrastre, are often a second time subjected to the patio process of amalgamation. This second treatment of the residues is found to consume a large quantity of quicksilver, and to afford only about one-half the total amount of silver which they contain; although some azogueros are of opinion that the loss of quicksilver is considerably reduced, and the yield of silver proportionately increased, by mixing them with from three to four times their weight of crude ore, as usually prepared for the patio. It is, however, probable that this mixture of fresh ores may only have the effect of so complicating the calculation of the result obtained, as to render it difficult to estimate, with any great degree of accuracy, the produce of the roasted residues.

Instead of effecting the washing in circular cisterns provided with agitators, this operation, in the districts situated nearest to the city of Mexico, is usually conducted in a wooden tank, which at one extremity is pierced with holes, which are placed at different heights from the bottom, and admit of being closed by plugs, whilst a constant stream of water is admitted at the other end. After this cistern has become nearly filled with water, the lama to be washed is thrown into it, and briskly stirred with shovels. When the ore has thus become well incorporated with water, the holes are successively opened; thus drawing off, at first, the lighter earthy matters, and subsequently the heavier metallic sulphides, until the amalgam remains in the bottom, in a state of considerable purity. After escaping from this box, the slimes pass for a distance of from sixty to eighty feet over inclined riffle boxes, by which a portion of the amalgam and mercury, which would otherwise be lost, is retained. This method of washing is generally considered more expensive, and to be attended with a greater waste of amalgam, than those previously described.

Filtration.—When the amalgam has been purified from the last adhering particles of mineral, by wiping with flannel, it is filtered through a cone-shaped bag, called a manga, Fig. 43, of which the upper portion is covered with leather, whilst the lower consists of strong, closely-woven canvas. This is hung by chains or cords from a stout beam, and when the mixture of mercury and amalgam is introduced, its weight causes a large portion of the quicksilver to escape through the meshes of the sail-cloth in a liquid form, and to fall into a vessel placed beneath it for that purpose. The amalgam, which is allowed to remain in the manga during a greater or less time, in accordance with the quantity operated on, finally assumes the appearance of fine sand, and is then ready for conversion into bricks, previous to distillation. This amalgam usually contains mercury to the amount of from five to five and a half times the weight of silver present. The proportions differ, however, in accordance with the quantities operated on; the relative amount of silver being greater when the weight of the mass is very considerable. The amalgam at the upper part of the filter is also always more argentiferous than that lying near the middle of the bag, whilst that at its lower extremity frequently does not contain above one-seventh part of that metal. In some cases the manga is made sufficiently large to contain from two to three thousand pounds of amalgam at a time; and when the weather is dry, and large quantities of mercury have passed through it, the apparatus frequently becomes highly electric, giving off sparks very freely to any conducting body that may approach it.*

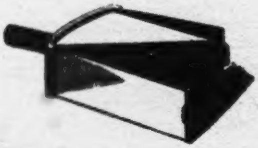
The filtration of a charge usually occupies about two hours; and when the mercury has ceased to drip from the bottom of the bag, the manga is emptied on a table covered with leather, and the amalgam beaten into triangular bricks in iron moulds having the form shown, Fig. 44.

FIG. 43.



MANGA, OR STRAINER.
(*From Tilmann.*)

FIG. 44.



MOULD CONTAINING BRICK OF AMALGAM.
(*From Tilmann.*)

These bricks, or *bollos*, are each from three to four inches in thickness, and so shaped, that when six of them are placed together, they form a circular cake, leaving a round hole in the centre for the escape of mercurial vapours during the process of retorting.

Retorting.—This operation, by which the separation of mercury and silver is effected, is conducted by the aid of a large iron or copper bell, which is placed over the amalgam, and around which is kindled a charcoal fire. A circular tank of masonry is constructed below the floor of the burning-house, through which a stream of water is constantly caused to flow; and in this is placed an iron tripod, covered by a round plate, having a hole in its centre for the escape of mercury. On this plate are piled the bollos of silver, to such a height as to reach to within a short distance of the top of the bell, which, when placed over them, leaves a space of about an inch between its sides and the column of amalgam. When thus arranged, the bell (*capellina*) is lowered over it, and the bottom secured, either by lute or a water-joint, constantly supplied by means of a pipe. Unburnt bricks, or *adobes*, are now built around the arrangement in the form of a hollow wall, leaving an annular space between them and the bell, of about eight inches in width. This is filled with charcoal, which is ignited, and as the temperature increases, the mercury becomes volatilised, and, passing into the chamber below the floor, is condensed, collects in a liquid form, and escapes by an iron pipe into a proper receptacle. The fire is thus kept up during about fifteen hours; after which the apparatus is allowed to cool, and, when sufficiently cold, the bell is removed, either by a windlass or by means of simple blocks, as shown, Fig. 45.

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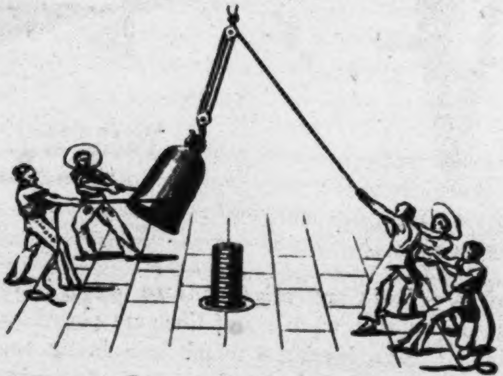
* Duport, p. 269.

Fig. 45.

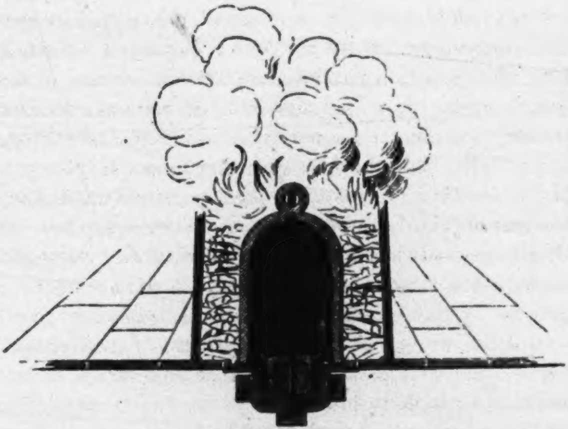


Lifting the Capellina.
(*From Tilmann.*)

This silver, which is found to have assumed a porous structure, and a beautiful frosted appearance, is called *plata piña*, and is placed in leathern bags for removal to the smelting house, where it is assayed, and run into bars. The silver obtained by the patio process of amalgamation is in most cases very nearly pure, being generally above 990 fine; and in many cases, as at Guanaxuato, almost absolutely pure silver is obtained.

In some localities, the arrangements for retorting by the capellina are slightly varied from those above described, as shown, Fig. 46, in

FIG. 46.



CAPELLINA.
(*From Tilmann.*)

which the amalgam is supported beneath the bell B, on a stand A, enclosed in a cast iron vessel C, kept cool by means of a current of water constantly flowing beneath the bottom, and through the annular cavity D. The condensed mercury escapes, as soon as deposited, by means of a wrought iron pipe, into a proper receiving vessel. In some cases, the charcoal is retained in its place by means of a circular iron grating.

The interior measurements of the bell are usually as follow: height, 3 feet; diameter, 18 inches; thickness of metal, 1½ inches. The charge of amalgam is about 2,000 lbs., affording 400 lbs. of silver; the consumption of charcoal per charge is 500 lbs.

Loss of Silver.—The loss of silver by this process of amalgamation is considerable, but varies in different localities, in accordance with the nature of the ores operated on, and the degree of fineness to which they are reduced by grinding. At Guanaxuato, the average loss may be estimated, on docile ores, at from 10 to 14 per cent. At Fresnillo, the results of assays made during a year on ores containing a considerable quantity of galena, pyrites, and blende, as compared with those actually obtained from the patio, showed a deficit equal to 28 per cent. of the assay produce. According to Duport, the loss experienced on the ores from the Veta Grande, at Zacatecas, which contain a large amount of antimonial sulphides of silver, averaged from 35 to 40 per cent. of the produce, as indicated by assay. A portion of this loss is, however, mechanical, occasioned by particles of amalgam being carried off by the water employed for washing the lamas, after amalgamation; and a more efficient system of separation would, consequently, have had the effect of reducing the amount of silver so carried away. In some establishments, and particularly at Guadalupe-y-Calvo, the sulphides resulting from the ordinary process of washing in the lavadero are subsequently treated on shaking tables, and the results obtained are stated to be satisfactory.

Loss of Mercury.—It has been long established as a principle among Mexican azogueros, that, in order to obtain a given amount of silver by the process of patio amalgamation, it is absolutely necessary to sacrifice an equal weight of mercury. To this deficit of quicksilver they apply the name of *consumido;* whilst any excess of loss above this amount, and which is considered to be due to want of care, and mechanical causes, is called *perdida.* The total loss of this metal may be consequently divided into two elements, of which one is constant and the other variable, and to which different names are applied.

All calculations are made, in Mexico, with reference to the production of a mark of silver weighing eight Spanish ounces (3,550·5 gr.); so that, in case of the total deficit of mercury being twelve ounces per mark of silver, an azoguero would divide the loss as follows:—

Consumido	8 ounces	3,550·50 gr.
Perdida	4 „	1,775·25 „

It is, however, evident that this principle is not of universal application; since metallic silver, which occurs in considerable quantities in certain ores, combines directly with quicksilver, forming an amalgam of which the weight corresponds with the united amounts of the two metals weighed separately. The total loss of mercury varies in accordance with the nature of the ores, the method employed for washing the contents of the torta, and the greater or less proportion of native silver present. In many cases this loss does not exceed ten ounces per mark of silver, whilst in others as much as twenty-four ounces of quicksilver are expended for each mark of silver obtained.

The average loss may probably be taken to vary from ten to sixteen ounces per mark of silver extracted. The time necessary for working a torta varies, according to circumstances, from fifteen to forty-five days.

Fine Stamping.—At Real del Monte, where the Freiberg process of amalgamation in barrels is extensively employed, the ores are for this purpose reduced by wet stamping. The arrastre, although the slimy nature of the ore ground by it is well suited for patio amalgamation, yields a bad return for the mechanical power employed, and the slimes produced are not well adapted for the barrel process, for which fine sands are found preferable. The great augmentation of weight and bulk arising from the gradual wearing of the grinding-stones is also an objection to this apparatus, and it has consequently been found advantageous to grind the ores destined to be treated by the Freiberg process in ordinary stamping mills. Into these the ore is gradually fed by a hopper, a small stream of water being at the same time introduced, which, displaced by the successive falls of the heads, carries with it all the finer particles of ground ore which float over an inclined plane, whose height regulates the fineness of the grinding. The ground ore is subsequently conducted into large tanks, in which it becomes deposited; the water finally passing to a pump, by which it is elevated and returned to the stamps coffer, to again commence its duty as a sorter and carrier of the reduced ore. Thirty stampers, when driven with a velocity of from sixty to eighty blows per minute, grind weekly 100 tons of hard quartz to an exceedingly fine sand. The consumption of stamp heads, where such mills are employed, is necessarily very great, and to effect the annual reduction of 35,000 tons of ore at Real del Monte, not less than 60 tons of cast iron are expended; but as new heads can be obtained at iron works in the immediate vicinity of the mines, they are readily replaced at a moderate cost. In some localities in which water power is abundant, the ores are sufficiently reduced by the stamping mill to fit them for treatment in the torta without a preliminary grinding in arrastres.

Roasting.—The minerals destined for being worked by patio amalgamation are, in Mexico, seldom subjected to any preliminary treatment, with the exception of grinding; but in the case of highly-pyritous ores, they are sometimes partially roasted, for the purpose of removing the excess of sulphur. At Zacatecas, certain kinds of ore, after being broken into small lumps, are roasted in heaps, by mixing them with wood and covering the pile, which is surrounded by a wall of loose stones, with a layer of charcoal. This operation occupies but a few hours, and, after its completion, the ores are ground in arrastres, in the usual way. In the districts of Tasco, Sultepec, &c., where metallic sulphides are particularly abundant, the ores are, after grinding, generally roasted in the reverberatory furnaces employed in the preparation of magistral. The fuel employed for this purpose is wood; but although the roasting of a charge frequently occupies as much as twelve hours, but a comparatively small proportion of the sulphur is thus eliminated. The *marmajas*, or concentrated sulphides, obtained by washing on the planilla, are also roasted in the same way.

Chemical Reactions of the Patio.—The general opinion entertained by the various authors who had, previous to Sonneschmid, written on the subject of patio amalgamation, appears to have been, that in the different argentiferous ores, the silver was covered by various substances, such as sulphur, arsenic, and antimony, and that this covering prevented its forming an amalgam with mercury. The salt added to the torta was supposed to possess the property of removing these impurities from the surface of the silver; but this "clearing action" was thought only to be developed in presence of a sufficient amount of magistral, properly moistened, and which produced this effect through the agency of the heat produced. They also believed that for each mark of silver extracted an equal amount of mercury must necessarily be lost, and that any further expenditure of quicksilver in the progress of the operation was due to mechanical causes.

To Sonneschmid, who published his work entitled "*Tratado de la Amalgamacion de Nueva España*," in 1825, belongs the credit of first presenting a rational explanation of the nature of the reactions which take place during the process of patio amalgamation. This author, who, from his knowledge of chemistry and his long practical experience, was well fitted for the task which he undertook, refuted by unanswerable arguments the notions which had hitherto been entertained; and by means of a close examination on the one hand, of the various phenomena which present themselves during the progress of the operation, and by the aid of the light then recently thrown on the subject by the progress of chemical science on the other, propounded a theory which, with slight modifications arising from a more advanced state of chemical knowledge, is that generally entertained at the present day. According to Sonneschmid, that portion of the silver which exists in the ores in a native state is alone capable of uniting directly with mercury; and if, in grinding with this metal any ores which do not contain silver in the metallic form, a small quantity of amalgam be obtained, it is produced by the action of some substance which in presence of mercury has the property of reducing the silver existing in a state of combination. These compounds, as well as the native metals, are susceptible of conversion into "muriate of silver," under the influence of "muriatic acid" liberated by the action of the sulphuric acid of the magistral on a solution of common salt. The muriate of silver thus formed may be destroyed by the addition of alkaline earths, but the silver will then be converted into an oxide which has no longer the property of forming an amalgam with mercury. Further, that as certain metals have the peculiarity of separating others in a state of purity from the acids with which they are combined; mercury performs this part with regard to silver, by taking from it the muriatic acid by which a portion of it is destroyed, whilst the remainder forms an amalgam with the liberated silver. This reduction of silver by the action of muriatic acid on metallic mercury, together with the direct action of the same on that metal, are the two causes occasioning the loss of quicksilver; the direct action of the acid manifesting itself whenever it becomes necessary to make a further addition of magistral. The mercury lost remains in the residue, either in combination with muriatic acid, or in the metallic state; the former representing the deficit known as *consumido*, and the latter forming that portion of the loss classed as *perdida*.

It will be observed that according to this theory, the salt and sulphate of copper act only in furnishing the acids which they are respectively supposed to contain, and it remained for future observers, aided by a further development of chemical science, to point out the influence exercised by the sulphate of copper, as well as that of the common salt, as a solvent for the chloride of silver formed. Karsten first called attention to the fact that the addition of magistral caused the production of chloride of copper, by the aid of which the transformation of the sulphides of silver into chloride is chiefly effected, but without supporting his statement by any confirmatory experiment. The same chemist also pointed out the importance of the presence of a solution of salt as a solvent for chloride of silver, which is thus brought into intimate contact with mercury, at the expense of a portion of which it is decomposed, thus liberating the silver, previous to its entering into combination with the remainder. About the same period, Boussingault confirmed the statements of Karsten with regard to the formation of chloride of copper, and proved, beyond dispute, that this salt is abundantly produced during the process of patio amalgamation. The whole of the mercury by which the decomposition of chloride of silver is either directly or indirectly produced, appears to be converted into calomel, since no traces of the higher chloride have ever been discovered among the products of amalgamation by the patio.

The essential ingredients constituting a torta are salt, magistral, mercury, and sulphide of silver. Salt and the sulphate of copper in the magistral react on each other, giving rise to the production of chlorides of iron and copper, and sulphate of soda. The chloride of copper in its turn acts on the sulphide of silver, producing chloride of silver, which is dissolved in the excess of salt added to the torta; and the silver, finally reduced to the metallic state by a portion of the mercury which is ultimately converted into calomel, whilst the reduced silver enters into combination with the unattacked mercury, forming an amalgam. It has also been shown that the lower chloride of copper, formed by the action of sulphide of silver on the higher chloride, is dissolved in the solution of salt, and acts on another portion of sulphide of silver, also converting it into chloride, which is subsequently reduced by the mercury, and finally converted into amalgam. Boussingault has further proved that the copper of the magistral is ultimately to a great extent transformed into sulphide; sulphide of mercury is likewise occasionally found in the torta, and has, by some chemists, been supposed to be the result of the action of calomel on sulphide of silver, by which sulphide of mercury on the one hand, and chloride of silver on the other, would be produced. It is, however, probable that this substance may, in many cases, have been formed by the direct decomposition of sulphide of silver by metallic mercury, since it is well known that under certain conditions this action takes place. Messrs. Bowring and Uslar, both of whom were practically acquainted with the patio process of amalgamation, have contended that chloride of silver is not necessarily formed during its operation; but the various phenomena which they have brought forward in support of this view of the question, appear to be far from conclusive, and are all easily explained by the chloride theory, which is now almost universally received.*

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* Some chemists are of opinion that the chlorination of the mercury is, at least partially, effected by the higher chlorides of copper and iron, and that the lower chlorides of these metals, in their turn, assist in the reduction of chloride of silver.

Cost and Results of Patio Amalgamation.—The results obtained by this process and the cost of the various operations will evidently depend, not only on the nature of the ores, but be also, more or less, influenced by numerous local circumstances connected with each particular district in which it is employed. Under such circumstances it would be impossible to furnish the various items of expense under any general heading, and we shall therefore give the results of this operation as obtained from the books of some of the most important establishments in which it is employed.

Table *showing the result of* 11 *tortas of Ore from the Mines of Cata and Secho, reduced in the hacienda of San Joaquin, Guanaxuato, Mexico.*

No.	Date, 1868.	Weight of Lama.		Salt.	Magistral.	Mercury.	Silver produced.	Mercury.		Total loss of Mercury.	Days in Operation.	Number of Treadings.
								Perdida.	Consumido.			
		mon.	qtls.	ar.	ar.	lbs.	marks.	lbs.	lbs.	lbs.		
1	May 13.	81	20	573	95	3,200	870	1	461	462	15	8
2	June 9.	80	02	480	117	3,100	848	196	436	632	21	12
3	July 2.	72	21	432	109	2,721	780	100	402	502	23	10
4	„ 23.	66	30	379	93	2,450	634	43	335	378	17	9
5	Aug. 14	67	29	306	84	1,936	553	11	292	303	22	10
6	Sept. 20	75	29	329	70	2,529	719	73	379	452	25	9
7	Oct. 9.	77	07	293	68	1,711	508	143	276	419	36	15
8	„ 30.	57	18	285	48	1,920	566	117	299	416	30	14
9	Nov. 25	79	00	480	60	2,839	816	151	427	578	32	15
10	Dec. 23.	80	12	400	56	2,548	736	96	403	499	38	16
11	Jan. 14.	44	28	270	33	2,238 *	1,251	31	361	392	32	15
„	„ 24.	36	30	216	32	1,818		36	293	329	38	17
		821	02	4,443	865	29,010	8,281	998	4,364	5,362	27†	12½†

* One torta divided into two parts.
† Average.

Statement, *continued, showing the result of* 11 *tortas worked in* 1863, *at the Reduction Works of San Joaquin, Guanaxuato, Mexico.*

No.	Quantities.				Costs.				Produce.			Remarks.
	Ores.		Containing		Ores.	Working.	Raspa.*	Total.	Silver.	Gold.	Total Value.	
			Silver.	Gold.								
No.	cargas.	arrs.	marks.	grains.	dollars.	dollars.	dollars.	dollars.	marks.	grains.	dollars.	
1	770	13	1,028·58	38,817	4,816·54	3,182·88	………	7,099·42	869·90		7,247·75	Two of the arrastres used for grinding these tortas were charged with 4 lbs. of silver amalgam, and 41½ per cent. of the gold was obtained.
2	748	11	1,019·81	35,672	4,798·54	3,184·16	………	7,982·70	848·26		7,061·23	
3	718	04	929·37	39,807	4,358·04	2,814·18	………	7,172·22	780·15		6,513·15	
4	632	02	843·92	32,909	4,124·19	2,545·61	………	6,669·80	633·62		5,291·51	Two of the arrastres used for grinding these tortas were charged with 19 lbs. of copper amalgam; and each time it was found free from copper, some more of the same amalgam was added. Gold obtained = 62 per cent.
5	621	02	662·86	16,702	2,705·52	2,420·17	………	5,125·69	553·00		4,705·52	
6	691	05	919·74	21,096	4,128·60	2,752·52	………	6,881·12	718·92		6,001·17	
7	716	04	739·58	21,530	2,992·60	2,729·39	………	5,721·99	507·83		4,238·61	
8	526	04	743·70	32,323	3,484·22	2,164·74	………	5,648·96	565·50		4,722·00	
9	744	00	1,016·41	41,358	4,814·55	3,206·97	………	8,021·52	816·41		6,682·40	
10	747	09	967·10	27,276	4,467·94	2,951·49	………	7,419·43	736·12		6,144·06	
11	780	13	1,545·80	48,202	8,248·03	3,304·94	………	11,552·97	1,251·50		10,443·59	
								1,057·28				
	7,687	10	10,400·67	350,292	48,936·77	31,257·05	1,057·28	81,253·10	8,281·30		69,071·59	The salt employed contained about 85 per cent. of chloride of sodium; and the copper ore, used for preparing the magistral, 20 per cent. of copper. Loss of mercury perdida 3·44 per cent. Total loss of mercury per mark of silver 10·36 oz.
	1st Raspa								145·98	47,443	2,538·24	*In the whole of these tortas copper amalgam was employed.*
	2nd ,,								177·26	42,065	2,626·66	
	3rd ,,								150·84	19,439	1,828·35	
	4th ,,								298·37	37,958	3,448·97	
	5th ,,								173·59	22,552	2,083·16	
	6th ,,								219·45	33,562	2,711·60	
	Sundries								9·16	392	82·36	
									9,464·95	203,411	84,390·93	
	Loss								944·72	155,881	...	The loss of silver was 9·08 per cent.
	Total								10,409·67	359,292		
	Total Cost of 11 tortas								...	...	81,253·10	
	Profit								...	...	3,137·83	

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* The *Raspa* is that portion of the precious metals which is obtained by grinding with mercury in the arrastre.

TABLE *showing the results obtained at the Hacienda Nueva, belonging to the Fresnillo Company, during the years* 1840, 1841; *and including the first nine months of the year* 1842.*

Date.	Montones.	Silver.		Value.	Cost of Treatment.
		marks.	oz.	dollars.	dollars.
1840	31,995	147,851	3	1,293,675·12	664,274·13
1841	35,291	222,022	0	1,942,692·50	731,346·90
9 months of 1842	28,324	167,377	3	1,464,552·50	504,460·50
	95,610	537,250	6	$4,700,920·12	$1,900,081·53

These 95,510 montones = 85,366 tons of ore, produced 537,250 marks of silver, about 3,974,000 oz. troy; which gives a mean produce obtained by the process, of 0·0014 of the weight of the ores operated on.

The cost of treatment, per monton of 2,000 lbs.; the cost per monton, exclusive of the value of mercury expended; the average loss of mercury; and the average yield of silver, were as follow:—

Total Cost of Treatment, including Mercury.		Total Cost of Treatment, exclusive of Mercury.	Ounces of Mercury lost, per mark of Silver.	Mean Produce of Silver per Monton.	
Date.	dollars.	dollars.	ounces.	marks.	oz.
1840	20·76	14·45	14·064	4	5·00
1841	20·72	13·46	12·312	6	2·25
9 months of 1842	17·02	11·75	11·875	5	7·25

Mean result of Patio Amalgamation at Real del Monte, 1864-5†:—

Regla hacienda, reducing yearly	51,300 cargas.
Loreto " " "	54,430 "
Total yearly	105,730 cargas.
Mean assay of ore thus reduced	15·5 marks per monton.
Mean produce	14·1, or 9·0 per cent. loss.

The mean cost of reducing these ores was—

Coarse crushing in dry stamps, and subsequent fine grinding in arrastres	1·9 dollars per monton.
Manipulation in Patio	4·5 " "
General expenses of management	1·2 " "
Repairs	1·2 " "
	8·8
Sulphate of copper (2 dols. per monton)	3·2
Salt (1·6 quintals per monton)	6·5
Quicksilver (11 oz. per mark, silver)	6·5
Total . . .	$25·0‡

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* From Duport. † Furnished by Mr. Buchan.

‡ At Loreto, to this must be added the cost animal power in grinding.

Ore from Ophir Mine, Comstock, worked at Ophir Works, Washoe. Average cost per ton of working during eighteen months, ending June 1864.

PATIO

From December 1*st*, 1862, *to June* 1*st*, 1864.*

	Labour.	Wood.	Salt.	Sulp. Copper.	Mercury.	Castings.	Material.	Horses.	Total.
	dollars.	dollars.	dollars.	dollars.	dollars.	dollars.	dollars.	dollars.	dollars.
Crushing .	2·68	1·85	...	...	...	0·85	0·22	...	5·60
Beneficiating	3·82	0·19	5·95	0·55	2·85	...	0·07	1·56	14·99
Washing up.	1·48	0·09	0·17	0·10	0·80	...	0·02	...	2·66
Total Cost .	7·98	2·13	6·12	0·65	3·65	0·85	0·31	1·56	23·25

AMALGAMATION BY HOT PROCESS.—This process for extracting silver from its ores is much less employed in Mexico than in some portions of South America, where the ores are more generally found suitable for this method of treatment. The only ores which can be advantageously worked by this process are such as contain a large proportion of native silver, or in which that metal occurs in the form of chloride, iodide, or bromide.

The ores worked in the *cazo* are almost invariably of the description known as colorados, which are, generally speaking, coloured red by an admixture of oxide of iron. These, after being roughly stamped in the usual way, are subsequently treated in the arrastre, but, as they are afterwards concentrated by washing, care is taken not to carry the grinding to such an extent as would cause a large portion of the finely-divided silver ore to be carried off in suspension by the water. This concentration is effected by means of the inclined plane, called a planilla, before mentioned as being employed for treating the residual matters resulting from the washings of the torta in the patio process of amalgamation. By this means the ore is reduced to about two per cent. of its original weight, and the lighter portions, thus removed by water, may, if found sufficiently rich, be subsequently treated by the patio process.

The cazo is a vessel formed either of curved blocks of stone, or of wooden staves, like those of a barrel, and of which the bottom is made of copper.

The dimensions are usually as follow:—diameter at top, 3 feet,

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* Furnished by Mr. W. W. Palmer.

3 inches; diameter at bottom, 2 feet; depth, 18 inches. The bottom, which, as before stated, is of copper, has, when new, a thickness of 2½ inches, but becomes gradually thinner by use. This is retained in its place by a groove running around the interior of the vessel, near its lower extremity; all the joints being carefully luted with clay, with which the sides of the apparatus are thickly plastered, and which is kept in its place by an exterior wall of unburnt bricks. The copper bottom of the cazo rests on the wall of the hearth, thus forming the roof of a fireplace which has neither fire-bars nor chimney; and is provided with but one opening, which serves, at the same time, for the introduction of the fire, and the escape of the products of combustion. The fire is lighted, and a sufficient amount of water poured into the vessel to form, with the ore subsequently added, a very liquid paste. When the temperature of this mixture has reached the boiling point, and it has become strongly agitated by ebullition, salt is added, in proportions varying from five to ten per cent. of the weight of the ore operated on. It is, however, necessary not to introduce the salt until the contents of the cazo are in active ebullition, since it would otherwise form, with the ore, a compact mass, adhering firmly to the copper bottom, from which it could not be removed without emptying the apparatus.

As soon as the salt has been added, a workman, who squats on the side of the cazo, continually stirs its contents by means of a wooden agitator, with which he constantly rubs the copper bottom; and at this stage the first addition of mercury is made. The amount of that metal employed is carefully regulated according to the richness of the ore under treatment, and should never exceed twice the weight of the silver it contains. At first, one-fourth only of this quantity is added, and, about a quarter of an hour after its introduction, the workman takes a sample, by means of a bullock's horn attached to a wooden handle, with which he scrapes the bottom of the boiler so as to obtain a small quantity of the heaviest portions of the ore and amalgam. By washing this in a horn spoon, he afterwards removes the lighter constituents of the mixture, and exposes the amalgam, which, if the operation is progressing favourably, should present the appearance of finely-divided granules, of a light lead-grey colour. The amalgam in this state is called *polvo*, and it is known by experience that it then contains one-third of its weight of silver, or, in other words, that about two parts of quicksilver are united with one part of silver.

More mercury is subsequently added, and other samples taken, until it is found that the hardness and state of division of the amalgam begin to change, when the operation is considered to be terminated; but, before stopping, the workman makes another trial, called a *prueva en crudo.* For this purpose he washes a portion of the amalgam in such a way as to entirely remove all traces of mineral, and, after adding a little clean mercury, and rubbing the mixture with his fingers, he observes whether the quicksilver thus added becomes solidified. Should this be the case, he introduces a further portion of mercury, and continues the operation; since he has thus learnt that this metal has not been added in sufficient quantity to extract the whole of the silver which the process is capable of affording. When, on the contrary, the mercury introduced retains its fluidity, the liquid contents of the cazo are dipped out into reservoirs, whence they are subsequently removed, to be added to the ingredients of a torta; whilst the solid deposit of ore, which contains the amalgam, is stored in wooden cisterns, from which it is afterwards taken, for the purpose of being washed in large bateas. Before washing this deposit, an amount of mercury is added to it nearly equal to that which has been employed in the cazo, in order to produce a less dry amalgam, which, becoming united in a mass, is no longer so liable to be carried off in suspension with the mineral and earthy residues. The exact consistence which should be given to this amalgam, in order to avoid loss, is a matter of considerable importance, since, if too dry, a portion of it is readily carried away by the water; if, on the other hand, the amalgam be subjected to the process of washing in a liquid state, it is liable to be projected over the edge of the batea by the oscillation necessary for the removal of the various metallic sulphides with which it is more or less mixed. The cazo, which is the apparatus described by Alonzo Barba as that employed in his time for the amalgamation of silver ores, by the aid of heat, has, in the district of Catorce, been much enlarged, and, under the name of *fondon*, is extensively employed in some important metallurgical establishments.

The diameter of the copper bottom of the fondon varies from 5 feet 6 inches to 7 feet 6 inches; and, instead of the necessary friction being produced by the action of a wooden stirrer, worked by hand, it is obtained by means of an upright shaft, provided with cross-arms, to which are attached rectangular blocks of copper, set in motion by a mule, harnessed to a prolongation of one arm. This apparatus may therefore be regarded as an arrastre, in which the paving and stone voladoras are both replaced by metallic copper; and beneath the bottom of which is a fireplace, similar in all respects in its construction to that built below the ordinary cazo. In one of the sides of the fondon is an orifice on a level with its bottom, which, during the working of a charge, is closed by a plug; the removal of which allows its contents, both liquid and solid, to be drawn out into tanks, in which the heavier matters are allowed to settle, previous to being washed.

The weight of the charge, which for the cazo seldom exceeds 100 lbs., is in the case of the fondon increased to from 1,200 to 1,500 lbs. The time necessary for working this amount is six hours, being the same as that employed to work 100 lbs. in the ordinary cazo. The fuel employed consists of the wood of the palm-tree, which, from its small density, burns rapidly, and produces a large amount of flame. The amalgam obtained by washings is treated precisely like that resulting from the patio process; but the silver produced in this way invariably contains a little copper, which, at Catorce, is removed by cupellation with lead, in a furnace called a *galeme*.

The slimes separated from the amalgam by washing are, together with the residues from the cazo, treated by the patio process. In order to prevent adherence of the mercury or amalgam to the copper bottom of the apparatus, the workmen take care to always employ a less amount of quicksilver than would be necessary to form an amalgam with the whole of the silver present; but this is of comparatively little importance, since the whole of the residues of the operation are re-treated in the way above described.

By this process, the silver existing in the native state, as well as the chlorides, iodides, and bromides of that metal, is readily reduced; but this is not the case with regard to the sulphides, and it consequently becomes necessary, in order to obtain the silver which they contain, to have recourse to the supplementary treatment by the patio. In working such residues, however, the addition of magistral is not required, since they contain a sufficient amount of chloride of copper to convert the whole of the sulphides of silver into chloride.

The loss of mercury experienced during the treatment of silver ores in the cazo or fondon is extremely small, as at the close of the operation a weight of amalgam is obtained which, deducting the silver it contains, corresponds with the quantity of mercury originally employed; but this amalgam, in addition to silver and mercury, contains a little copper, and the total loss, which is purely mechanical, may be taken as being between two and three per cent.

By this process, the chlorides and other analogous compounds of silver are evidently not reduced, as in the patio, by the action of metallic mercury, but by the copper furnished by the bottom of the apparatus; and even if it be admitted that this reaction gives rise to the production of the higher chloride of copper, which has, under certain circumstances, a chlorinising action on mercury, this could not take place in the presence of the excess of metallic copper furnished by the bottom, which would at once transform into the lower chloride any of the higher chloride of copper which might be produced. If, in treating ores by this process, the quantity of mercury added be so great as to be attended with the adherence of the amalgam to the copper bottom, the operation is found to proceed very slowly, and a great loss of mercury is the result; as, under these circumstances, the chlorides, iodides, or bromides, present in the ore, being cut off from direct contact with the metallic copper, are reduced at the expense of quicksilver. The most important condition necessary for the economical working of the cazo is to keep its bottom constantly free from any adherence of quicksilver or amalgam; and when the amount of mercury added does not exceed twice the weight of the silver present, there is no danger of any inconvenience arising from this cause. With even twice this amount of mercury, Duport states, he was enabled to work without inconvenience; but the moment that proportion was exceeded, the amalgam attached itself firmly to the copper plate, and a large loss of mercury was the result.

Although this process of amalgamation is only adapted for the treatment of colorados, which contain chloride and other analogous salts of silver, it has sometimes been applied to the reduction of the negros, in which the silver exists in the form of sulphides. Under these circumstances, it becomes necessary to make an addition of magistral, which causes a most destructive action on the mercury present, and which is, apparently, not modified by the presence of the bottom of metallic copper, since this loss of quicksilver often exceeds four times the weight of the silver obtained.

At Catorce, the poorer ores, after being previously concentrated by the planilla, were, according to Duport, in 1843, treated in the cazo, at a total cost of one dollar six reals per carga of 300 lbs.

The richer class of ores, on the other hand, which it was not necessary to wash, but which required fine grinding and great care in working, was operated on in the fondon at an expense of two and a half dollars per carga.

The loss of mercury was estimated at two per cent., and the silver obtained contained about one per cent. of copper.

ESTUFA AMALGAMATION.—In some of the colder and more humid districts of Mexico, a modification of the patio process has been employed. The ground ore, instead of being exposed in the open air, on a paved court-yard, as in the ordinary patio process, is placed under a shed, and the usual method of patio amalgamation proceeded with, until the operation is about half completed. The ore is then removed into a sort of room, termed an *estufa*, or stove, which has under it a fireplace, six or eight feet long, so connected by side flues with small chimneys as to elevate the temperature of the room containing the ore. Here it is exposed to a gentle heat, and allowed to remain during two or three days, when it is again removed, and the reduction completed by the ordinary method of patio amalgamation. By this process, the time required for the reduction of the ore is less than by the patio, and the yield of silver greater; the loss of mercury, on the other hand, is more considerable.

CHAPTER XVII.

TREATMENT OF SILVER ORES BY AMALGAMATION—BARREL PROCESS.

FREIBERG—BARREL AMALGAMATION—WHEN INTRODUCED—COMPOSITION OF ORES—CHLORINATION—AMALGAMATION—DISTILLATION OF AMALGAM—REFINING—CONSTANTE—GRINDING CRUDE ORES—CALCINING WITH SALT—SIFTING AND GRINDING—AMALGAMATION AT CONSTANTE—TREATMENT OF AMALGAM—MELTING AND REFINING—TREATMENT OF RESIDUES—COST OF TREATING ORES AT CONSTANTE—COST AT REAL DEL MONTE—THE BARREL PROCESS IN NEVADA—METHOD OF CONDUCTING, AND COST OF OPERATION.

THE first works erected in Europe for the treatment of silver ores by amalgamation appear to have been those described by Schlüter as having been put up at Kongsberg, for the purpose of treating the stamped ores of that district. This apparatus consisted of eighteen small cylindrical vessels, arranged in a circle, in which the ores were mixed with mercury and kept in a state of constant agitation by means of a vertical spindle in each tub, the whole being worked by a large horizontal toothed wheel placed in the centre. It was not, however, until the latter part of the century that the attention of mining engineers on the continent of Europe became particularly directed to the process of amalgamation. In the year 1780, the Baron de Born suggested to the Austrian Government the propriety of adopting this system in the mining districts of Hungary; and, at his solicitation, some experiments were instituted at the Glashütte Works near Schemnitz. Although these trials were not followed by the introduction of this system of amalgamation into the Hungarian mines, they, nevertheless, gave rise to many valuable discoveries, of which the Saxon Government, which had deputed a commission to attend on De Born, availed itself in the erection of the works at Freiberg.

FREIBERG.—The first works were erected at Freiberg in the year 1790, under the direction of M. de Charpentier, but these were destroyed by fire shortly after their completion. The existing amalgamation works, commenced immediately after the destruction of the former, were completed in 1794, and finally closed about the year 1856.*

The amalgamation of silver ores was perhaps at one time more economically conducted at the Halsbrücke works, in the vicinity of Freiberg, than in any other European establishment.

Composition of Ores.—The usual constituents of the ores treated, were sulphur, antimony, arsenic, silver, copper, lead, iron, and zinc, which were more or less mixed with various earthy minerals, besides sometimes containing traces of bismuth, gold, nickel, and cobalt. In selecting these ores, they were so assorted as not to contain above four per cent. of lead or one per cent. of copper, as from combining with the mercury added, these metals gave to the amalgam produced a spongy consistency, and thereby rendered the treatment difficult and expensive. The different ores selected for amalgamation varied in produce from 15 to 200 oz. of silver per ton, and formerly the mixtures were so arranged that the charges of the furnaces should contain from 75 to 80 oz. per ton. Latterly, however, it was usual to work the poorer and richer ores separately, as it was found that the loss of silver in the residues was thereby considerably diminished.

The mixtures of the poorer ores afforded, on an average, from 30 to 40 oz. per ton, whilst the amount of silver in those of the richer ores varied from 90 to 130 oz. per ton. It is essential that all mixtures of ores should contain a certain proportion of sulphide of iron for the formation of sulphates, the presence of which is necessary to the success of the operation of roasting: the quantity of sulphide of iron present should, generally speaking, be about twenty-five per cent. If the amount of pyrites naturally occurring in the ores was not equal to this proportion, addition was made either of that mineral, or of crude sulphate of iron. Sometimes, however, the ores at Freiberg contained more iron pyrites than was necessary to the success of the operation, and in such cases it was found advantageous to subject the more sulphurous of them to a preliminary roasting without addition of salt, and thus reduce the average amount of sulphur in the mixture to the right proportion.

Chlorination.—The ore, properly selected, was laid on a large floor forty feet in length by twelve in width, and on the top of it was thrown about ten per cent. of common salt, let down from an upper room by means of spouts placed in the floor for that purpose. The heap thus made up of alternate strata of ore and common salt, was well mixed, by being carefully turned over, and then passed through a coarse sieve. It was subsequently divided into small parcels or charges, weighing from 4½ to 5 cwt. each. The salt employed for this purpose was supplied by the Prussian salt mines. The mixture of ore and salt was now roasted in reverberatory furnaces, provided with fume flues for the reception of the pulverulent matters mechanically taken over by the draught. The prepared charge had now to be spread on the bottom of the hearth and gently heated, for the purpose of expelling the moisture, which, to a greater or less extent, it invariably contained. During the process of drying, usually occupying from twenty to thirty minutes, the charge required to be kept constantly stirred with a long iron rake. The lumps formed during this operation must also be broken down by means of a beater provided with a long iron handle. On the temperature being subsequently raised, white fumes were given off, and in about two hours from the commencement, the whole mass had become redhot. The charge was likewise occasionally turned, so that every particle of ore might be equally exposed to the action of the fire, and during the whole time the mass required to be diligently stirred with the rake. The fire was then left to burn down, and the combustion of the sulphur promoted by constant stirring. This went on without intermission, until the interior of the furnace had become dark red, and a sample taken from the mass no longer evolved any odour of sulphurous acid. During this process the ore increased considerably in volume, and the particles cohered so loosely together as to offer but little resistance to the movements of the rake.

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* The ores formerly treated at Freiberg by amalgamation are now, to a great extent, smelted with galena, and the silver extracted by crystallisation and cupellation.

After this the heat was again raised during about three-quarters of an hour; the sulphate of iron, formed by the oxidation of pyrites, thus reacted on the common salt, causing, in the presence of peroxide of iron, the evolution of chlorine and hydrochloric acid gases, which, coming in contact with the sulphides of silver, rapidly converted them into chlorides. Chlorides of the other metals present were at the same time formed, together with sulphate of soda. When this roasting was terminated, the charge was raked from the furnace into an iron barrow and removed to an adjoining floor. The ore was subsequently raised to an upper story for the purpose of being passed through a set of sieves, with the view of separating the finer powder from the agglutinated lumps. The latter were broken down to a proper size, and a portion re-roasted, by adding a small quantity to each of the subsequent ordinary charges. The remainder was mixed with from two to three per cent. of salt, and calcined in the usual way. The finer particles, which passed through the sieves, were, on the contrary, ground between heavy millstones, by which they became reduced to the state of an impalpable powder.*

After roasting, the ores, besides various earthy salts, chiefly consisted of peroxide of iron, basic sulphate of iron, the chlorides of iron and copper, and a certain proportion of oxide and sulphate of copper, sulphate of lead, oxides of antimony and zinc, and a small quantity of various metallic sulphides in addition to sulphate of soda, and the excess of common salt employed. The different compounds of silver originally present in the ores were thus converted into the chloride of that metal, with the exception of traces of metallic silver, and perhaps, also, of minute quantities of sulphide and oxide of silver. The charge suffered in roasting a considerable diminution in weight, generally amounting to about ten per cent.; this loss being chiefly due to the escape of sulphur, chlorine, and the volatilisation of common salt, zinc, antimony, arsenic, and the chlorides of iron and copper.

Amalgamation.—The amalgamation of the roasted ores was at Freiberg conducted in twenty wooden casks, arranged in four rows, and turning on cast iron axles secured to the ends by means of bolts. These barrels, which were internally two feet eight inches in length, two feet eight inches in diameter at the ends, and two feet ten inches in the middle, were made of pine wood three and a half inches in thickness, and strengthened by iron hoops and binders.† On one of the ends of each tun was placed a toothed wheel working in a pinion keyed on a shaft receiving its motion directly from a water-wheel. The general arrangement of these barrels is represented, Figs. 47, 48.

Above each of the tuns so arranged, was placed a wooden box C, Fig. 47, into which was thrown the roasted ore.

At the bottom of this was a spout terminating in a hose *h*, made of strong canvas, finishing with a cylinder of tin plate, for the purpose of introducing the powdered ore into the different barrels B. Each cask was furnished with a circular opening *a*, five inches in diameter, fitted with a wooden plug, through which was bored a small hole

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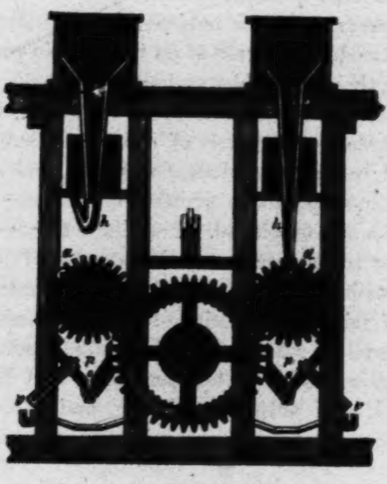
* The crude ores were not crushed at the works, but at the mines, where they were often enriched by mechanical treatment.

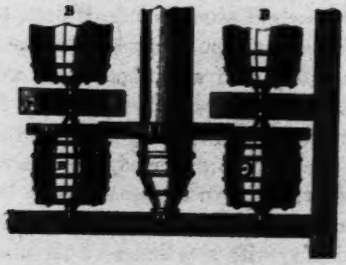
† At Constante, and at the different works in Nevada, the barrels are perfectly cylindrical in form.

Fig. 47.



Amalgamating Barrels.
(*Sectional Elevation.*)

Fig. 48.



Amalgamating Barrels.
(*Plan.*)

provided with a pin, made of hard wood, for the purpose of running off the argentiferous mercury at the termination of the process.

Below the tuns, and a little above the level of the floor, were placed triangular troughs destined to receive the residual matters at the termination of the operation. Before charging the barrels, 3 cwt. of water were run from the vessels E, exactly holding that quantity, into each; after which 10 cwt. of finely-ground and sifted ore were introduced through the hose *h*. Each cask should also contain from 80 to 100 lbs. of wrought iron, cut into fragments of about an inch square and three-eighths of an inch in thickness, which, in proportion as they become dissolved by the action of the substances with which they are associated, require to be replaced by fresh pieces. As soon as the barrels had been charged, and the bungs firmly secured in their places by binding screws, the apparatus was thrown into gear, by a screw acting on a sliding pillow-block, and made to rotate with a rapidity of from twelve to fifteen revolutions per minute. At the expiration of two hours the barrels were in succession stopped for the purpose of examining the state of the metalliferous paste which they contained. If the charge was too firm, a little water was added; but if, on the contrary, it was too soft, a small quantity of ore was introduced. When this had been attended to, five cwt. of mercury were drawn off into each cask, and the tuns, after being securely closed, again thrown into gear and kept constantly revolving for about sixteen hours, at the uniform rate of thirteen turns per minute.

During the first eight hours of this period they were, at Freiberg, twice examined for the purpose of ascertaining whether the paste was of the proper consistence; for if too thick, the mercury becomes very finely divided, and if too thin it remains at the bottom, and does not become sufficiently incorporated with the various constituents of the charge. In the first case it is necessary to add a small quantity of water, and in the second a little roasted ore. After the introduction of mercury, the temperature becomes considerably raised by the chemical changes constantly going on, so that, even in winter, it sometimes reaches 104° Fahr.

At the expiration of eighteen hours the amalgamation was in this establishment ordinarily complete, and the tuns were then filled with water, and again made to revolve, during from one and a half to two hours, with a velocity of only six or eight revolutions per minute. By this means the mercury was separated from the slimes with which it was mixed, and became collected in one mass at the bottom of the tuns. When this union of the globules of mercury had been accomplished, the different casks were successively thrown out of gear, and stopped with the apertures uppermost. The small peg in the bung was now removed, and in its place was inserted an iron pipe, to which was attached a small hose with a screw and clasp, for the purpose of closing it when required. The cask was then so turned that the plug should be immediately over the spout *o*. The hose being put into the iron tube *p*, the mercury was allowed to run off into the gutter *v*, by which it was conducted to a receiver prepared for that purpose. This period of the operation was closely watched by the workmen, who, the moment any of the earthy matters began to flow from the orifice, at once closed it tightly. The barrels were now turned with the apertures *a* upwards, the small hose plug removed, and the large bung loosened by a few taps with a mallet. The barrels were subsequently turned mouth downwards, the bungs withdrawn, and the muddy residuum discharged into troughs situated immediately under them, from whence it flowed into large washing vats, or tinas, placed on the ground-floor below the barrels. In the course of fourteen days 180 tons of mineral were treated in this establishment, every ton of which required an expenditure of 3 lbs. of metallic iron, and 8·95 oz. of mercury, so that every pound of metallic silver produced was obtained at an expense of $\frac{35}{100}$ths of an ounce of quicksilver.* The loss of silver experienced amounted, on an average, to from seven to nine per cent. of the total amount present, as indicated by assay, but the latter figure may be taken as most nearly approximating to the mean result.

During the first two hours that the casks were set in motion, before the introduction of mercury; the chlorides contained in the roasted ore were thus reduced to the state of minimum chlorination, the saline matters dissolved by the water present, and the particles of chloride of silver exposed. If, instead of this, the mercury had been immediately introduced into the barrels, it would, by reacting on the sesquichloride of iron, &c. have become partially converted into calomel, which not being again reduced, during the subsequent stages of the operation, would have resulted in a considerable loss of that metal. This loss is, however, avoided by the action of metallic iron, since the protochlorides thus formed are without action on metallic quicksilver. The chloride of silver contained in the roasted ores is, in the Freiberg process, decomposed by agitation with metallic iron, the chlorine combining with it to form protochloride of that metal, whilst the reduced metallic silver becomes subsequently dissolved in mercury. The chlorides of lead and copper which may be also present, are reduced at the same time as the chloride of silver, and enter into the composition of the amalgam produced.

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* Winkler states the loss of mercury at Freiberg to have averaged during five years 1·41 loth, or $\frac{3}{4}$ ounce nearly per centner of ore, assaying 3 to $3\frac{1}{2}$ oz. of silver; and 3·57 loth, or $1\frac{3}{4}$ oz. per mark of silver produced.

Subsequently, however, a great reduction in the loss of mercury was effected. In 1853 the amalgamation master stated the loss was then scarcely $\frac{1}{3}$ oz. per mark. We cannot, however, vouch for the accuracy of this assertion, and our own experience would lead us to somewhat doubt its correctness.

Treatment of Residues.—The residues conducted to the washing vats before mentioned, were mixed with an additional quantity of water, and kept constantly stirred by agitators attached to iron arms worked by an upright shaft in the centre of each vat, which received its motion from a small water-wheel. These vats were furnished with openings, at various distances from the bottom, through which the tailings, held in suspension by the water, could be successively drawn off into tanks, in which they were allowed to settle. When these residues contained more than four and a half ounces of silver per ton, they were removed to a drying floor, and subsequently re-roasted with from fifteen to sixteen per cent. of iron pyrites, and from five to six per cent. of common salt. These calcined residues were sifted in the usual way, and then, without being re-ground, subjected to amalgamation in barrels for a somewhat shorter period than was customary in the case of ordinary ores.

The mercury collected in the bottom of the different washing vats was drawn off every five or six weeks, and from the large proportion of base metals which it contained, it was treated apart from the ordinary amalgam produced in the usual manner. The mercury and amalgam obtained by tapping the barrels, were afterwards filtered through close canvas bags in the ordinary way. The amalgam which was retained in these bags consisted of a mixture of six parts of mercury and one of an alloy composed of about eighty per cent. of silver, and twenty of a mixture of copper, antimony, zinc, lead, and some other metals.

Distillation of Amalgam.—The amalgam, after being well pressed, was subsequently heated in iron retorts placed in suitable furnaces, and the mercury separated by distillation from the non-volatile metals which were obtained in the solid form. Three retorts were latterly employed for this purpose, and into each were introduced, on iron dishes, 350 lbs. of amalgam; usually resulting in the production of about fifteen per cent. of retorted silver at the close of the distillation, which generally occupied ten hours. The base metals contained in the alloy of silver thus produced, were, with the exception of a certain proportion of copper, removed by a process of refining in a large iron crucible, which was conducted in the following way.

Refining.—The crucible was first placed in the furnace and made red hot, when the lumps of silver were successively introduced and brought to a state of fusion. Powdered charcoal was then thrown on the surface of the fused metal, and the crucible temporarily covered with a thin plate of iron. This, after the lapse of a few minutes, was again removed; and the impurities which had risen to the surface, were, together with the unconsumed charcoal, skimmed off by means of a perforated ladle. More powdered charcoal was then thrown on the fluid metal, and the scum removed as before. These operations were repeated, with occasional stirrings of the metallic bath, until the surface of the metal had become bright and clean. This process occupied from six to eight hours, and when completed the metal should be malleable and dissolve completely in nitric acid, to which the addition of an excess of ammonia should impart a clear blue colour free from turbidity. The silver was afterwards cast into ingots of a semi-hemispherical form, and in that state forwarded to the Saxon Mint. The dust removed from the furnace flues was, at Freiberg, after being sifted, mixed with, and treated as, ordinary silver ore. The slags and sweepings from the various melting operations, were crushed and afterwards fused with carbonate of soda and nitre, by which means the silver was obtained in the metallic state. The water run off from the tanks in which the residues from the tinas were allowed to settle, contained, in solution, a considerable amount of common salt and sulphate of soda, together with small quantities of sulphate of iron, and various other soluble salts.

Barrel Amalgamation at Constante, Spain.—The ores treated at Constante are obtained from the mines of Hiendelaencina, and contain silver in various states of combination, but principally in the form of antimonial sulphides disseminated in gangues, chiefly consisting of sulphate of baryta, together with a small proportion of quartz, accompanied by considerable quantities of carbonate of iron, which chiefly occurs in the richer portions of the veins.

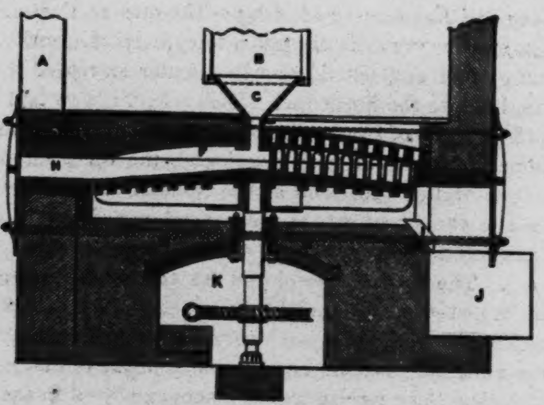
In addition to the sulphides of antimony, arsenic, &c., in combination with sulphide of silver, the ores contain iron pyrites and uncombined sulphide of antimony, with small quantities of galena and sulphide of copper. The proportions of these constituents vary, however, with almost every parcel of ore received into the works. The mineral, as delivered at the establishment, is likewise more or less mixed with micaceous slate derived from the rocks in which the lode is enclosed; and the poorer ores usually contain much more of this substance than the richer varieties, to being mixed with which, they generally owe their impoverishment. Immense quantities of such ores, too poor to be amalgamated with profit, were formerly wasted at the mines; but this loss might, to a certain extent, have been avoided by employing a better system of extraction, and greater care in the selection of the ore.

Grinding and Roasting Crude Ores.—The ores as they are brought from the mines are immediately taken to cylindrical Cornish crushing mills, and ground and sifted through circular sieves of wire gauze, having ten holes to the linear inch. Each parcel is now laid apart and weighed in presence of two parties; one of whom is appointed by the mines from which the ores are purchased, the other on the part of the reduction works. As soon as the mineral has been weighed, it is removed to the stores where the several parcels are piled one on another heterogeneously, and from whence the ore is taken as it may be required. The furnaces employed for the calcination of the ores with salt are, at Constante, eight in number; six of them having revolving hearths worked by machinery similar to Brunton's calciners, whilst the remaining two are hand furnaces of the ordinary reverberatory description. The mechanical calciners employed in the establishment make from three to four revolutions per hour, and have movable hearths fourteen feet in diameter. Each of these furnaces requires an expenditure of about half a horse power to turn it, and is provided with a single fireplace consuming from 120 to 140 lbs. of pine wood per hour. An improvement on this apparatus has been effected by Mr. W. West, who employs two fireplaces, and thus more evenly distributes the heat over the surface of the hearth.

This improved calciner is represented, Figs. 49, 50, in which A is the doorway of the calciner house; B, a window in ditto; C, cast iron hopper through which the ore is fed; and D, three cast iron fluke or agitator frames built in the masonry of the arch. The revolving table E, on which is spread the mixture to be calcined, has a slightly conical surface, and is made of the best firebricks set on end in a cast iron shell. The lining of the furnace F, in which the calcining table revolves, should also be built of refractory material. The fireplaces G, are situated on either side of the furnace, after traversing which, the products of combustion make their escape by the flue, H, to the chimney.

After calcination, the ore passes through the cast iron spout I, into the chambers J, in which it is deposited. The opening K, beneath the furnace, is for the purpose of allowing space for working the worm wheel and pinion communicating motion to the revolving table. The foot-block, carrying the perpendicular shaft, is supported by the block of stone L, and the furnace is tightly braced by the girders M.

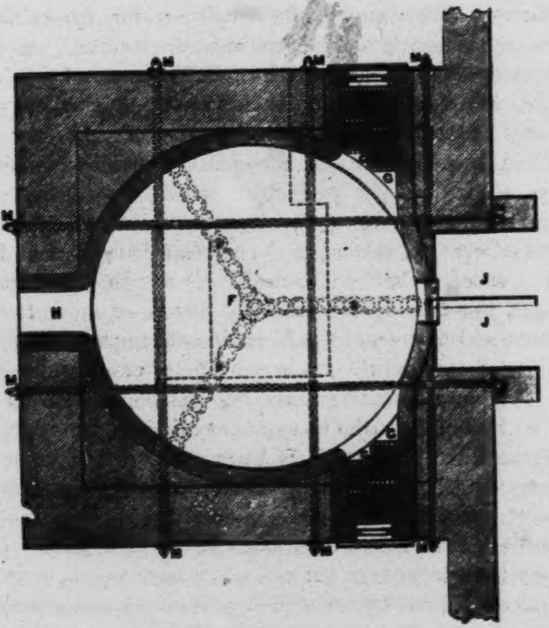
FIG. 49.



REVOLVING CALCINER.
(Vertical Section.)

The mixture of mineral and salt is made as intimate as possible, especially for the mechanical furnaces, since in them the charge is but little disturbed by the rakes: they are charged with ore and salt by means of the iron hoppers placed immediately over the centre of each of the hearths. For the supply of the hopper a heap of about fourteen quintals of ore, with from five to six per cent. of salt, is, from time to time, prepared upon the platform on the top of the furnace, and a few shovelfuls thrown in occasionally, as required; taking care, however, always to have enough in the hoppers to prevent the escape of acid vapours through them from the furnace. The time during which the mineral remains in the apparatus, and the quantity calcined per hour, must necessarily depend on the rapidity of motion given to the revolving hearth, and the angle at which the iron stirrers are fixed. The average amount passed through each furnace in twenty-four hours is about eighty-four quintals, or three and a half quintals per hour; or for every revolution of the bed, nearly one quintal is discharged from the furnace. Compared with the German roasting furnace, the mechanical furnaces are found less efficient for the treatment of rich ores; particularly when they are charged damp and contain much sulphur, in which case the excessive production of lumps becomes a serious inconvenience. But in the treatment of the class usually brought to the works, they possess the advantage of calcining a larger quantity in a given time, and they require no further attendance than is necessary for supplying them with ore and fuel. The management of the fires is also a matter of importance, since should they be forgotten,

FIG. 50.



REVOLVING CALCINER.
(*Horizontal Section.*)

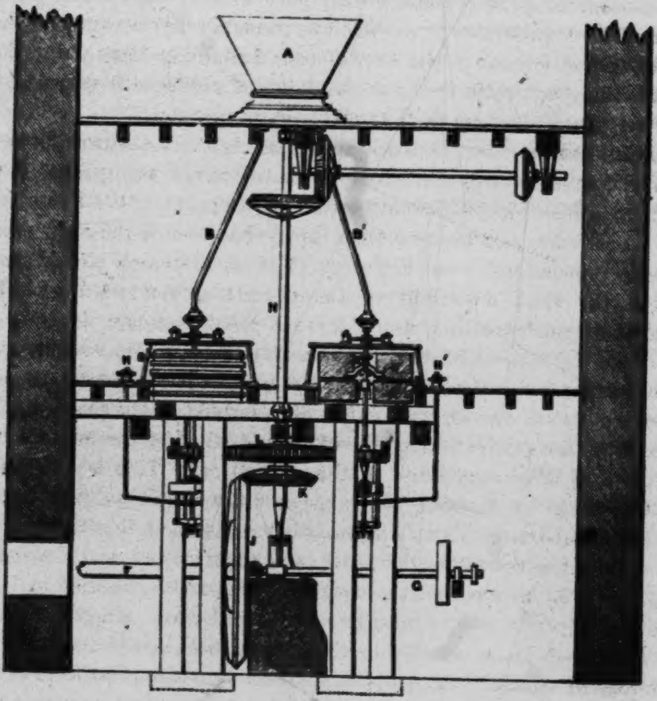
and the heat become lowered, the mineral, from continuing to pass at the same rate through the furnace, cannot be properly calcined; and in order to raise the temperature, after having been neglected, the workmen sometimes charge the grate with fuel to such an extent as to overheat the ore.

Of the three hand furnaces used at Constante one is very old, and only employed for drying salt before grinding, and for calcining sweepings, and other refuse matters, containing silver. The other two are similar in form to those of Freiberg, and are worked in much the same manner. They are chiefly employed for the calcination of such ores as contain 9 oz. and upwards per quintal. A charge for these furnaces weighs about 540 lbs., and consists of about 495 lbs. of dry ore, 40 lbs. salt, and 5 lbs. moisture. A smaller proportion of salt may be employed in the calcination of very poor ores, but it is seldom less than six per cent.; for the richer minerals it is necessary to add a larger proportion of salt, frequently as much as 15 per cent. When a charge is introduced into the furnace, it is evenly spread over the surface of the hearth by means of an iron rake, and for the space of half an hour scarcely any fuel is thrown on the grate, whilst the mineral is constantly stirred, in order to evaporate the moisture and to prevent the formation of lumps. The fire is then gradually increased, and the ore, which by degrees becomes so heated that the sulphurous matters enter into ignition, is stirred uninterruptedly with the rake, so as to expose continual fresh surfaces to the action of the air. The charge is likewise turned, in order to, from time to time, equally expose every part to the action of the fire. After the lapse of about two hours the mineral has become fully heated, when the fire may be left to burn down for about an hour. It is then again urged for nearly three quarters of an hour longer, so as to cause the sulphates produced by the oxidation of metallic sulphides to react on the salt, which, in the presence of metallic oxides, evolves chlorine, which combines with the silver and other metals present. The whole operation, from the introduction to the withdrawal of the charge, occupies four hours, during which time from 140 to 160 lbs. of wood are consumed. The metallic sulphides, contained in the rich ores operated on in the furnaces, do not amount to more than from 9 to 10 per cent., whilst in the poorer ores the proportion is often considerably less.

Sulphide of iron is a necessary ingredient in silver ores to be submitted to this process of amalgamation, and during the roasting process is oxidised and partially converted into sulphate which, reacting on the salt, evolves the gases necessary for converting the silver into chloride. The proportion of sulphide of iron requisite for this purpose appears to be variable as regards different ores. At Freiberg at least 20 per cent. was considered indispensable, whereas at Constante good results have been repeatedly obtained with only 8 per cent., even when operating on rich minerals. The silver ores of that neighbourhood are but little accompanied by sulphides, and arsenides of other metals, and the pyrites found is very poor in silver and easily oxidisable. The comparative purity of these minerals may account for their ready calcination, for in the case of ores containing large proportions of sulphides, arsenides, &c., a great excess of chlorine is required for the purpose of effecting their complete decompostion.

In the hand furnace, as much as 14 per cent. of salt was formerly added, but with ores containing 9 oz. of silver per quintal 6 per cent. has sometimes been found sufficient. Separating the richer from the poorer ores, and treating them apart, has been found very advantageous here, as well as at Freiberg. The calcination of the ores with salt is the most important of the operations connected with this process of amalgamation, as on it, in a great measure, depends the nature of the results obtained from the barrels; since, in whatever way the operation may be conducted, the silver extracted will be commensurate with the amount of chloride formed in the furnace. The success of the calcination is, therefore, judged of by the proportion of chloride of silver contained in the roasted ore. This is ascertained at Constante by digesting for some hours a small weighed portion of the roasted mineral with warm dilute ammonia; it is then thrown on a filter, the residue well washed, and finally dried and assayed for silver. It is obvious that the weight of this metal obtained indicates the quantity remaining uncombined with chlorine. Another method, in some respects preferable to the process with ammonia, is to treat the roasted mineral with a strong solution of common salt. For this purpose a few grammes of the calcined ore, just as it is drawn from the furnace, are thrown on a filter upon which is poured a hot saturated solution of salt, until the liquor, which filters through, no longer affords a white precipitate of chloride of silver on being largely diluted with cold water. The residual ore is then dried and assayed for silver, as in the former case. It has been suggested that the crude mineral, which is at present sifted through an eight or ten-hole sieve, would be better if ground a little finer, but a certain degree of coarseness is necessary in order that the oxygen of the air and the gases evolved in the process of calcination with salt may become freely diffused throughout the charge. If the mineral be very finely ground previous to calcining, the porosity of the charge is greatly diminished, and consequently the calcination is found to proceed more slowly; besides which the formation of lumps is thereby facilitated.

Fig. 51.



Horizontal Mills.
(*Constante.*)

Sifting and Grinding Roasted Ores.—The calcined ores before being submitted to amalgamation in the barrels are first sifted through a circular sieve of sixty holes to the linear inch, and the portion which does not pass through is then very finely ground, in order that every particle of silver which it contains may be exposed to the action of mercury; the portion passing through the sieve, amounting to 40 per cent. is taken directly to the barrels for amalgamation. The coarser portion of the ore, which is allowed to accumulate in the furnace-house, is, when required, removed and passed between a pair of cylinder mills from which it falls into a revolving sieve of sixty holes to the linear inch; the coarser ore escaping from this sieve is then raised by an elevator, and dropped into the hopper of a second pair of smooth cylinders, from whence the mineral falls into another sieve of the same degree of fineness as the first. The hard coarse particles, which refuse to go through this second sieve, are finally ground in horizontal mills of French burr stones. The arrangement of four of these mills, of which there are eight in the establishment, is represented, Fig. 51, in which one pair of stones is shown in section. Into the circular hopper A is introduced the stuff to be ground; B B′ are small pipes of sheet iron for delivering the stuff between the surfaces of the runner C, and the bedstone C′; D casing enclosing the runner into which the ground material is delivered; E hole in centre of runner; F driving shaft with continuation G for belt pulley; H H′ regulating screws for elevating the runners C; J driving wheel; K crown wheel; L wheel giving motion to pinions M M′; N vertical shaft for driving supplementary apparatus. Four pairs of stones are driven by the wheel L. The surface of the runner is parallel with the bedstone from the periphery to within one-third of its diameter; the line of the lower face of the runner then feathers upwards in order to receive the feed.

The following particulars afford sundry details relative to this apparatus:—

Diameter of stones	4 ft. 2 inches.
Thickness of bedstone	12 inches.
„ runner	14 „
Gauge of stuff in hopper, about . . .	100 holes to square inch.
„ „ on delivery	3,600 „ „
Number of revolutions	100 per minute.
Quantity of stuff ground per 10 hours .	1 ton per pair of stones.
Horse-power employed, about . . .	5 „ „
Revolutions of sizing sieves	23 per minute.
Diameter „ „ . . .	30 inches.
Length „ „ . . .	108 „
No. of holes per sq. inch in sizing sieve	3,600.
Character of runner	French burr.
„ bedstone	Compact quartz, moderately hard.
Duration of runner	Average, 18 weeks.
„ bedstone	„ 22 „
When dressed	Every third day.

In place of fine sifting and carrying the remainder of the calcined ore directly to the mills, and there crushing and grinding the whole to a fine powder, it might be desirable to first remove the hard lumps of ore, which are always produced in the operation of roasting, by passing the mineral through a sieve of about the same fineness as that employed in the mills for grinding the crude ores.

The lumps, which would be separated from the calcined mineral by the sieve above mentioned, are found to contain a considerable proportion of silver not converted into chloride, which is owing to the circumstance of these lumps being chiefly formed at the commencement of the operation, so that the inner portions escape the action of the gases evolved at a later period. These lumps should, therefore, be crushed, and submitted to a second roasting, either alone or mixed with the ordinary charges of crude ore. In the hand furnaces a much smaller proportion of lumps is produced than in the mechanical ones, owing to the constant stirring to which the mineral is subjected in the former.

Amalgamation.—The total number of barrels employed is sixty, each being, internally, three feet four inches long, and two feet six inches in diameter. The barrel department is divided into three sections:—No. 1, contains twelve barrels; No. 2, contains twenty-four barrels; and No. 3, twenty-four barrels. All of them can be driven by water power; those of Nos. 1 and 3 can likewise be worked by steam. The barrels are placed at a height of about twelve feet from the ground-floor. Immediately over them, and projecting through the floor above, are fixed iron hoppers for receiving the mineral to be amalgamated, which is introduced into the barrels by means of leathern hose, with which the whole of the hoppers are provided. The average weight of a charge for each barrel is thirteen quintals; but the hoppers are capable of holding a still larger amount. These latter are charged by gangs of men, each of whom carries a sack of mineral, weighing about a quintal. In the barrel-house No. 3 a railway is laid down between the hoppers, by means of which they can be regularly and quickly filled with the requisite quantity of ore. The internal shape of the barrels is, when new, cylindrical; but, after being in use for some time, their dimensions are increased by the wearing away of the inside, near the middle.*

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* In the amalgamation works of Nevada the barrels are made of 2-inch planks, and internally lined with 4-inch blocks of pine, similar to an internal wooden pavement. These blocks are so arranged as to cause the wear to take place across the

The amount of water introduced into each barrel varies with the nature of the ore, but the usual quantity is about 320 lbs.; and the iron contained in each amounts to 100 or 150 lbs. Considerable wear and tear of the barrels is caused by using large scrap iron of every kind for this purpose; and its action is less effective than would be the case with iron in smaller pieces.

The charge of ore and water having been introduced, the barrels are set in motion, and made to revolve at the rate of from eight to ten revolutions per minute during two and a half hours, in order that the ore, water, and pieces of iron may become completely incorporated. They are then stopped, and the state of the pasty mass examined. If this be too soft, or too stiff, more mineral, or additional water, is added, as the case may require. Should the paste be found of the right consistency—that is, just sufficiently stiff to allow of being formed into a ball in the hand—the mercury is poured in; the weight of that metal introduced being four quintals to each barrel. The loss of quicksilver is rather diminished than augmented by employing large quantities, since its fluidity is less affected by its combination with silver. After the introduction of mercury, the barrels are driven at the rate of from eighteen to twenty turns per minute, for a space of sixteen and a half hours, at the end of which time the amalgamation is considered to be accomplished. The barrels are now filled with water, for the purpose of rendering the paste liquid, and collecting the globules of argentiferous mercury into one mass for drawing off, and kept slowly revolving during a further space of two hours. In discharging the barrels, the mercury is first drawn off into the trough fixed beneath each row of casks, and thence conducted to an iron vessel, in which is collected the quicksilver run off from all the barrels; and any earthy matters which may escape with the mercury are carefully cleaned off the surface with flannel cloths. The fluid metal is subsequently run into canvas bags, which retain the solid amalgam, whilst the more liquid mercury filters through, and is received into an iron tank, situated beneath. During this operation, which lasts upwards of an hour, the barrels are continued in motion; because the whole of the residues having to pass through the vessel in which the quicksilver is collected, before they can arrive at the washing vats, cannot safely be discharged until the mercury has been run into the filters.

grain, and whenever a lining has become worn out, it can be readily replaced; the adoption of this form of construction has resulted in a considerable economy.

The process of amalgamation includes three operations:—

1. The barrels are charged with ore, water, and iron (without any mercury), and made to revolve, at the rate of ten turns per minute, for two hours, or until the mixture has become thoroughly incorporated, and is of the consistency of thick paste. This being accomplished, the mercury is introduced.

2. The casks are immediately afterwards put in more rapid motion, viz. eighteen to twenty turns per minute, and continued at this speed for about sixteen and a half hours; at the end of which period the complete amalgamation of the silver is supposed to have been effected.

3. In order to collect the argentiferous mercury into a mass for drawing off, it is necessary to reduce the consistency of the paste contained in the barrels. For this purpose they are filled with water, and driven slowly, at eight or ten turns per minute, for the space of two or two and a half hours. The amalgam is then discharged, and subsequently the residues; the barrels are discharged every twenty-four hours. The average richness of the ores treated may be taken at about 4¼ oz. per quintal.

Treatment of Amalgam.—The solid amalgam collected in the canvas bags above mentioned is treated in a very simple form of distilling apparatus, for the purpose of separating the mercury from the silver. The amalgam to be distilled is moulded into a cylindrical form upon an iron tripod supporting a perforated disc, depending from which, and fixed in brickwork, is an iron tube, dipping into a vessel of water placed beneath. Over the cylinder-shaped pile of amalgam is placed either a copper or cast iron bell, and perfect contact with the plate is ensured by carefully luting the joint. A circle is then made with loose bricks, raised around the bell at a short distance from it, and the space between filled with charcoal. The heat afforded by its combustion volatilises the mercury, which is quickly condensed and is collected in the cold water supplied to the cistern beneath. About 2,000 lbs. weight of amalgam are usually operated on at a time, nearly forty-eight hours being required to complete the distillation; and as many as forty sacks, weighing, in the aggregate, eighty arrobas, of charcoal are consumed during the operation.*

As may be supposed from the form of the apparatus, a portion of amalgam fuses, and falls into the cistern below; which, however, may be readily separated by filtration from the fluid mercury. This generally amounts to 60 or 80 lbs. in weight. The quantity of crude silver yielded by 2,000 lbs. of amalgam is generally about 290 lbs., or 14½ per cent. of the amalgam operated on. This silver has a brownish-white colour, is exceedingly porous, and may be readily broken in pieces under the hammer.

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* This apparatus is merely a modification of the Mexican capellina.

Melting and Refining.—The porous silver obtained from the distillation of amalgam is melted, at Constante, in a species of cupola, called a *cras.* When the metal is uncontaminated by impurities which impair the malleability of the resulting bars, the cras may still be employed for melting it, although a loss of silver must result from exposing it, at so high a temperature, to the action of a powerful blast. But when, as is most frequently the case in amalgamation works, the silver to be melted contains matters which impair its malleability, the removal of these is readily effected by this apparatus. The amount of impurities required to impart the property of brittleness to fused silver is, nevertheless, very small, and chiefly consists of lead, sulphur, antimony, and iron. These bodies may be readily got rid of by cupellation with lead, an operation, however, which, for this purpose, is attended with considerable trouble and expense. Melting the silver in black-lead crucibles appears to be, therefore, preferable. It may be thus purified by repeated additions of powdered charcoal, skimming and stirring, and be finally obtained in a perfectly malleable state. Instead of employing black-lead crucibles, large iron pots, capable of melting from 500 to 600 marks at a time, were used at Freiberg, and found to answer very satisfactorily. By fusion in large wrought iron crucibles, the silver may be obtained free from all impurities impairing its malleability, and rendering it unfit for the coining press.

Treatment of Residues.—The residues, on being discharged from the barrels, are conducted into large vats, in which revolve upright stirrers, fixed on four radial iron arms. The mud is retained in these vats for several hours, and kept in continual movement by the stirrers; in order that the particles of amalgam, disseminated in the residual matters, may subside to the bottom, and be afterwards collected, a constant stream of water flows into the vats during the whole time of washing. On being discharged from the vats, the mud flows directly into the river, in the bed of which it accumulates until it is carried off by a freshet.

COST OF TREATING ORES AT CONSTANTE.

Cost of working Mechanical Furnaces—

Wood for each furnace per week	940 rs.*
Salt, 6 per cent. 3,528 lbs. at 14 rs. per quintal	494
Splitting wood	19
Labour for each furnace	184
	Rs. 1,637

Ore calcined per week, 588 quintals in each furnace; 1637÷588=2·8 rs. per quintal.

Hand Furnaces—

Each furnace calcines 210 quintals of rich ore per week.

" " consumes 294 arrobas wood	294·0 rs.
" " " 12 per cent. salt=2,520 lbs.	353·0
Labour	242·6
	Rs. 889·6

889·6÷210=4·23 rs. per quintal.

Hence cost of calcining in mechanical furnaces is	2·8 rs. per quintal.
" " " hand "	4·23 " "

Average cost of roasting, 3·2 rs.

Section of 24 Barrels—

Wear and tear	800 rs.
Iron consumed, 5 arrobas	120 "
Mercury, reckoning the loss at 5½ oz. per mark of silver, or 454 lbs. at 5·62 rs.	2,551
Labour	1,157
	Rs. 4,628

Ore amalgamated per week by 24 barrels,
2,352 quintals; 4,628÷2,352=1·96 rs.

Incidentals	·04
Cost of amalgamating	2·00 rs. per quintal.

Summary.

Cost of crushing, per quintal	0·95 rs.
" calcining " "	3·20
" amalgamating " "	2·00
" distilling amalgam, and melting into bars	0·80
Sundry labour and materials	3·00
Total, exclusive of salaries, &c.	Rs. 9·95

or about 45s. 6d. per ton for ores containing on an average 4½ oz. of silver per quintal.

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* Reals Vellon = 2½d.

The quantity of ore which 60 barrels will reduce per week is 5,880 quintals.

Weight of mineral which all the furnaces will calcine in the same time	4,058	„
Viz. Six Mechanical Furnaces	3,528	„
Three Hand „	530	„

These works have consequently the means of reducing upwards of 1,800 quintals of mineral per week more than the present number of furnaces can, when in full work, calcine. Forty barrels would therefore be apparently sufficient to amalgamate all the mineral that can be roasted, but, as the supply of water is irregular, it is advantageous to have extra barrels, so that whilst the calcination is carried on without interruption, the ore accumulated during the dry season may be readily reduced when the supply of water is plentiful.

Notes relative to cost, &c. at Constante:—

Cost of firewood per 100 lbs., 10*d.*
„ mercury per lb., 1*s.* 2*d.* including carriage.
„ salt „ 100 lbs., 3*s.* „ „
Average cost of carriage of ore from mines to work, about 3*s.* per ton.
Cost of ordinary labour per diem, 1*s.*
Estimated loss of mercury per mark of silver, 5½ oz.
„ „ silver per cent. on assay produce, about 12.
The length of time a wooden barrel will last in regular work, 6 months.
Miller's wages at horizontal mills, 9½*d.* per ton of ore ground; cost, including wear and tear, 2*s.* 3*d.* per ton.
At Miner's crushing mill, usual cost, including labourers and wear and tear, about 3¾*d.* per ton.

The sifting of the calcined ore is done by contract at the rate of ¾*d.* per quintal of fine powder. This fine powder usually amounts to 35 or 40 per cent. of the calcined mineral.

REAL DEL MONTE, MEXICO; MEAN RESULTS OF BARREL AMALGAMATION, 1864-5.*

Velasco hacienda, reducing	107,700	cargas yearly.
Sanches „ „	48,550	„
St. Miguel „ „	49,600	„
Total reduced in the three haciendas	205,850	cargas yearly.

The mean assay of the ore thus reduced was	13·0	marks per monton.
The mean produce	11·3	„ „
Mean loss	1·7, or 13 per cent.	

The mean reduction cost in the three works was as follows:—

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* Furnished by Mr. Buchan.

Grinding by wet stamps	2·20	dollars	per monton.
Drying, sifting, and roasting	2·75	„	„
Amalgamation in barrels	1·10	„	„
General expenses of management . . .	1·20	„	„
Repairs to furnaces and machinery . .	1·55	„	,
Total . .	$8·80		
Fuel (wood, 21 quintals per monton) .	3·70	„	„
Salt (2 quintals per monton)	8·30	„	„
Quicksilver (4·7 oz. per mark of silver) .	2·80	„	„
Total per monton .	$23·60		

To this must be added, for auxiliary steam-power, Sanchez $2
„ „ Velasco $3

BARREL PROCESS AS CONDUCTED AT THE OPHIR WORKS, ETC., NEVADA.—*Drying the Ores.*—The ore is first dried on a kiln composed of twenty parallel flues 12 in. × 12 in., covered by cast iron plates, half an inch in thickness.

Crushing.—The rock is stamped dry, in batteries with four discharges in each coffer, through brass wire screens of 1,600 holes to the square inch.

Roasting.—It is then roasted, 1,300 lbs. at a charge, in reverberatory furnaces 8 ft. 6 in. × 9 ft. in the hearth; salt is added from twenty minutes to an hour and a half after the furnace is first charged. The time required to roast a charge varies from 4½ to 6 hours; about 4¾ hours is the average time necessary for second class ores, and the usual quantity of salt added is 5½ per cent. One hour before the charge is drawn, from 1½ to 8 per cent. of carbonate of lime is added in order to decompose a portion of the sulphates and chlorides of copper, zinc, &c., to prevent loss of quicksilver during the amalgamation, and also to produce an amalgam of greater purity. The ore when in the furnace is continually stirred, and is turned three times during the roasting: 1stly, two hours after the furnace is charged; 2ndly, before the limestone is added; and, 3rdly, when the limestone has been in about 35 minutes. Mr. G. Attwood, the former manager of these works, states that he found, that by a careful addition of limestone all the chlorides and sulphates of copper, &c. in excess, can be decomposed.

Screening.—The roasted ore is put through a bolt with a screen of 1,600 holes to the square inch, and then elevated to the barrels.

Amalgamation.—The barrels are charged with one ton of ore, and run with water and 450 lbs. of iron during three hours. It is found that with a large amount of iron in the barrels, upwards of 4 per cent. more silver is obtained than with the usual quantity, of 10 per cent., of that metal.

From 350 to 400 lbs. of mercury are now added, and the barrels allowed to run, at twelve revolutions per minute, during 12 or 13 hours, when they are filled up with water and drawn off, after being again run for two hours.

The amalgam obtained is strained through a stout canvas conical bag, and the tailings washed in a settler 15 ft. in diameter, passing from thence through a series of sluice boxes into a flume about 600 ft. long and four feet wide provided with riffles.

The amalgam yields about one-sixth of its weight of bullion, averaging 650 parts of silver in a thousand; the most impure samples assay about 375, and the best 992 thousandths.

Distillation.—The amalgam is distilled in circular retorts 4½ ft. long and 10 in. in diameter, set on an arch of brickwork, with three dampers to direct the flame towards any part where the heat may be required.

Returns from Ophir Ores, worked at the Ophir Company's Reduction Works, Nevada, for the Months of June, July, August, September, and October, 1865.*

Quantity. tons.	lbs.	Class.	Total Value.	
1,998	1,160	3rd	$121,512·71	
101	940	1st	49,450·51	
2,100	100		$170,963·22	$81·41 Average per Ton.
Returned 130 bars; weight 11,732 lbs. Valued at .			$158,774·20	$75·60 Return obtained.
Loss			$12,189·02	7·12 per cent.

Total loss of Quicksilver 2,176 lbs., or 2·96 oz. to each lb. of Bullion.

Returns from Ores extracted from Mexican Mine, Virginia City, and Empire Mine, Gold Hill, worked at the Mexican Company's Mill, at Empire City, in July and August, 1865.

Quantity. Tons.	lbs.	Value in Gold.	Per Ton.	Value in Silver.	Per Ton.
137	1,525	$7,585·17	$55·06	$12,753·80	$92·58
Bullion produced . .		$6,237·21	$45·27	$12,225·64	$88·74

Produce of Gold . . . . 82·25 per cent.
„ Silver . . . . 95·82 „ „
Value of Ore . . . . $20,338·97 = $147·64 per ton.
„ „ Bullion . . . . . . $18,462·85 = $134·02 „ „
Produce . . . . 90·74 per cent.

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* Communicated by Mr. G. Attwood.

Experiments made by Mr. G. Attwood in order to determine the loss experienced on the precious metals in the operations of roasting and amalgamation of Comstock Ores.

One thousand pounds of Black "base metal" ore, from the Ophir North Mine, roasted for 4½ hours with 5·50 per cent. of salt, and 2·50 per cent. of limestone, weighed 1,060 lbs. It lost 4·99 per cent. in value, and the mixture gained 3 lbs. in weight—

Assay value, silver	$43·43
" " gold	50·22
Total	$93·65

Again, 700 lbs. "base metal," North Mine ore, from the Ophir.
300 lbs. first class " " " "

Total 1,000 lbs., roasted 5½ hours with 7 per cent. salt, and 6 per cent. limestone, weighed 1,123 lbs., lost 6·72 per cent. in value, and 7 lbs. in weight. Twenty-one tons of the same ore, after being roasted, assayed

Silver, per ton	$99·97
Gold " . . .	92·80
	$192·77 = $4,048·17 value of parcel.
Value of bullion returned . . .	$3,915·57
Loss in amalgamation	3·27 per cent.

The loss experienced during barrel amalgamation is, however, generally greater than the above, as the samples were in this case taken out of the barrels before being drawn off, the amalgam washed out by hand, and the residue dried and assayed. The average loss on working the ores from the Comstock vein in barrels is 13·50 per cent. Ophir ores work better than Savage, Chollar Potosi, &c.

Composition of Ores.—The following analyses of ores from the Ophir Mine will serve to show the character of the rock usually treated at the different mills in the vicinity of Virginia City.

	1st Class.	2nd Class.	3rd Class.
Gangue . .	63·38	80·70	95·75
Silver . .	2·78	0·89	0·10
Gold . . .	0·05	0·03	0·00
Lead . .	4·15	4·04	0·40
Sulphur .	7·95	3·05	1·04
Zinc . . .	14·45	5·51	0·48
Copper . .	1·59	1·43	0·30
Iron . . .	5·46	3·46	1·55
Antimony .	0·08	trace.	trace.
	99·89	99·11	99·62

The analysis of an outside chip of a bar of bullion, obtained by barrel amalgamation, from the ores of the Comstock vein, gave the following results:—

Gold	1·58
Silver	41·51
Lead	39·01
Copper	17·04
Zinc	0·56
Iron	0·17
Total	99·87

Assay of samples from flues of Roasting Furnaces at the Ophir Reduction Works, August 5th, 1864.

	Value.
No. 1 contained Lead 28·02 per cent.	
„ „ Silver, per ton 60·14 oz.	$ 78·18
„ „ Gold „ 1·21 „	25·01
Value	$103·19
No. 2 contained Lead 24·66 per cent.	
„ „ Silver, per ton	$122·80
„ „ Gold „	62·63
Value	$185·43

COST OF REDUCTION OF ORES BY BARREL AMALGAMATION AT THE OPHIR WORKS, 1865.

Labour, of every description, per ton	$ 9·50
Wood, at $5 per cord	6·00
Salt, 4 per cent. at 3 cents. per lb.	2·40
Quicksilver, 1 lb. at 64 cents.	0·64
Shoes, Screens, Shovels, Belts, Tools, &c.	0·75
Castings, Scrap Iron, Timber, &c.	0·65
Charcoal, Assay office expenses, &c.	0·20
Total	$20·14

These ores assayed about $80·00 per ton. Ores assaying over that yield cost from $20·14 to $27 per ton, according to the amount of salt required, quicksilver consumed, and lime employed in furnaces.

Each furnace requires four men, two on each shift, as the charge is constantly stirred, and about a cord of wood is burnt in 24 hours.* The usual charge, for $80 ore, is about 1,300 lbs. which is roasted during 4½ hours.

The chief cost of the barrel process is roasting, labour, and dry stamping. The cost of amalgamation is only about $3 per ton.

The cost of labour in Nevada may be taken at $4 per diem.

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* The roasting is conducted in furnaces entirely constructed of ordinary red bricks.

CHAPTER XVIII.

TREATMENT OF SILVER ORES BY AMALGAMATION—PAN PROCESS.

STAMPING – COMMON PAN—VARNEY'S PAN—WHEELER'S PAN—HEPBURN AND PETERSON'S PAN—SEPARATORS—WORKING IN PANS—RETORTING—ARRANGEMENT OF REDUCTION WORKS—TREATMENT OF ROASTED ORES IN PANS.

SHORTLY after the discovery of silver mines in Nevada it became evident, that owing to the prevailing high prices of labour, firewood, and other materials, none of the processes employed in other countries for the reduction of silver ores, could be rendered available in that locality for the treatment of rock assaying from $20 to $80 per ton: it consequently became necessary to have recourse to some method of operating which would dispense with roasting, as practised in the ordinary barrel process, on the one hand; and the frequent manipulation, and great expenditure of time involved in working by the patio, on the other; and to endeavour, by the introduction of direct amalgamation, to work the poorer description of ores at a profit. Many deposits of ore of extraordinary richness have been discovered in the Comstock vein, and the ores from these it has been generally found most profitable either to ship to England, or to work by the barrel process. Amongst such ores may be instanced a lot of 80 tons, shipped from the California claim, which yielded on an average $2,200 per ton. But the main deposits vary in assay from $35 to $70 per ton, and it is the treatment of these ores which furnishes the largest proportion of the silver obtained from the district.

The silver ores of Nevada are worked but to a limited extent in reduction works belonging to the mines themselves, but more commonly on what is termed "custom work" at a fixed price per ton of ore of 2,000 lbs. The price charged varies according to whether or not a percentage amount of the assay value of the ore be guaranteed. If the ore be worked without guarantee, it is the interest of the millowner to pass as large a quantity as possible through the mill, consistently with the maintenance of a respectable character for the quality of the work done, since at any time when the supply of ore from the mines may slacken, an opportunity arises for working over the tailings, which always remain the property of the mill. As at the first working of the mines the mill power was insufficient for working the ore as rapidly as it was raised, its produce was only roughly estimated, and consequently not unfrequently overvalued, but a more satisfactory method is now generally adopted in the district. As soon as the ore is brought out of the mine it is deposited on the pile or floors in parcels of from two to three hundred tons, every tenth waggon-load being reserved for dry stamping, so as to allow of careful sampling. The crushed ore is forwarded with the parcel to which it belongs, a small charge being made for crushing. The small sample retained by the mine is assayed, and from it the gross assay value of the whole parcel is determined, and the charges by the mine against the mill made accordingly.

Ores at the mines are usually assorted into three classes. The first consists of those whose assay value is over $90 per ton of 2,000 lbs. As these contain but a small proportion of free silver, or silver in the metallic state, but have in their composition a considerable amount of sulphur, in combination with "rebellious metals," such as antimony, zinc, lead, copper, and iron, they are reserved for treatment by calcination and subsequent reduction in barrels by the Freiberg process.

The second class consists of ores of the assay value of from $40 to $90 per ton. The third class ranges from $20 to $40 per ton; the second and third classes are worked by the pan process. These ores usually contain, in value, about one-third gold to two-thirds silver. Ores of the first class are crushed by dry stamping, those of the second and third are crushed wet. The ores are prepared for stamping by being crushed into fragments of about a pound in weight by Blake's crushers.

Stamping.—For wet crushing, stamps are used of from seven to nine hundred pounds per head, including the stem, and are driven at the rate of seventy blows per minute. They are fed by an attendant whose duty it is to regulate the supply of ore, water, and quicksilver, when that metal is used in the battery for amalgamating the free gold present. Amalgamation in the battery requires careful attention, principally to avoid the too rapid addition of quicksilver, which should be supplied in very small quantities only.

To amalgamate the free gold in a battery, the quantity of quicksilver to be used is about one ounce weight to each ounce of gold present; this is sufficient to collect the gold and form a dry amalgam. If, therefore, a mill will stamp 24 tons of ore in 24 hours, and the ore contains an ounce of gold per ton, it will be necessary to put into the battery an ounce of quicksilver every hour. When, in addition to gold, the rock under treatment contains metallic silver, the amount of mercury added must be proportionately increased. More than 80 per cent. of the assay value of the gold in the ore may, by careful manipulation, be thus obtained. The gold amalgam accumulates in the corners and crevices of the battery box, between the dies, on the breast of the mortar, over which the crushed ore is washed into the settling cisterns, and is even found in considerable quantities adhering to the shoulders of the stamp shoes. The amalgam thus obtained is very hard and heavy, and is commonly so rich in gold, as to be worth as much as ten dollars per ounce. The crushed ore is taken off from the mortar by a supply of water, equal to the run of a three-quarter inch-pipe to each set of five stamps, through screens in the back and front of the box. These screens are made of thin Russian iron perforated with holes punched by sewing-needles, with the points cut off, and set in dies as closely as consistent with the maintenance of sufficient strength to bear the necessary concussion. The needles employed are usually those known as Nos. 5 and 6.

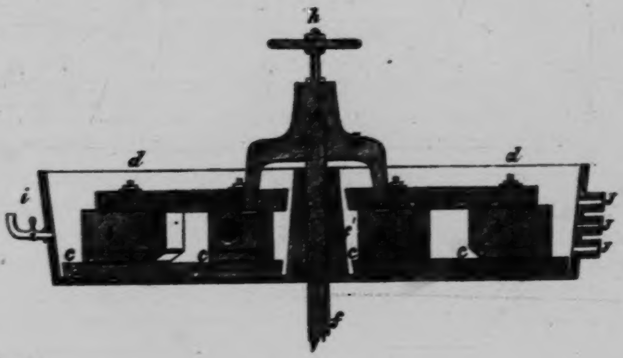
Settlers.—The troughs by which the crushed ore is conveyed to the settlers are provided with gates or stops in order to allow of the successive filling and emptying of the different tanks. These are made of wood usually about ten feet in length by eight in width, and three feet in depth. Here the ore is allowed to settle, and the water is run from tank to tank, and not allowed finally to escape until it has become tolerably clear.

As the water flowing from the settlers still contains much fine clayey matter, it is led off into one of a series of tanks which are successively filled, and after being again allowed to settle, the clear water is run off. This settling is much expedited by the use of two or three ounces of alum to each thousand gallons of water contained in the tank. It must be added in solution to each cistern of foul water, as soon as it has become filled, and sufficient time must be allowed for settling before the clear water is discharged. When the cistern is sufficiently full of mud it is run off, and the tailings dried for further treatment. Ores of the assay value of $80 per ton produce tailings of the value of $100 per ton, and in such quantities as to be equal to 20 per cent. of the total assay value of the ore.

Of the amalgamating pans employed in the reduction works of Nevada, it may be said that their modifications are almost endless. Much ingenuity has been expended on many of them, but the best are those of Varney and Wheeler, and that of Hepburn and Peterson. These makers have, however, all obtained patent rights, and supply their various amalgamators to the different reduction establishments at a fixed price; which includes the necessary licence for using them.

Common Pan.—There is, however, a more simple form of apparatus usually known as the common pan, with which results can, by careful working, be obtained almost as good as from those of the above makers. The common pan, Figs. 52 and 53, is a round wooden or cast iron tub, 6 feet in diameter, and about 2 feet in depth, with a flat bottom. A false bottom of 1½ inch iron is inserted into this, and a hollow pillar in the centre admits the passage of an upright shaft, which is generally worked by gearing, beneath the pan, capable of communicating to it from 15 to 20 revolutions per minute. To the wooden arms *a* are attached the blocks *b*, also of wood, to which are fastened the iron shoes *c*, by means of the bolts *d*, passing up through the arms. Each shoe has also an iron pin, about an inch in length, which fits into the wooden block and keeps the iron facing steadily in its place.

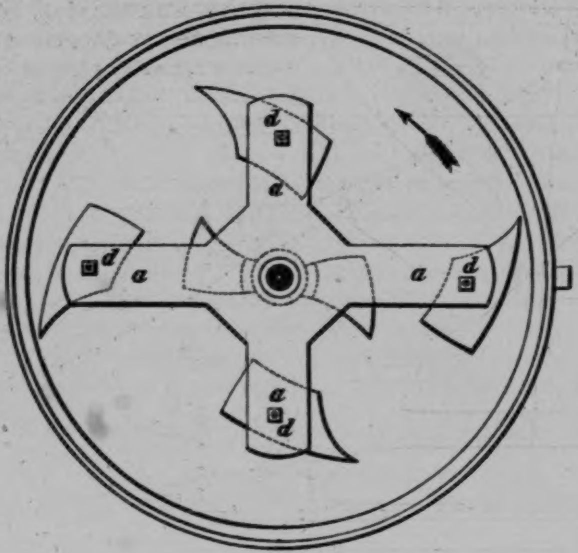
Fig. 52.



Common Pan.
(*Vertical Section.*)

On the shaft *f*, passing through the central pillar *f'*, is the yoke *g*, which, being fitted with a sliding key, can be raised by means of the screw *h*; and the ends of the yoke itself being attached to the wooden cross arms, the mullers will be raised at the same time. This arrangement for raising the mullers is not, however, very important, since they are usually allowed to grind with their full weight. Steam is introduced into the pulp by the pipe *i*, the discharge being effected by means of the apertures J. The false bottom is made one inch less in diameter than the bottom of the pan itself, and has an aperture in the centre an inch larger in diameter than the base of the pillar, in which the vertical shaft works. To fasten the bottom in its place, and prevent the mercury from finding its way under it, strips of cloth, about two inches in width, are lapped around the edge of the false bottom, as well as applied against the sides of the pan. A little iron cement is then poured in, and the bottom secured in its place by means of well-dried wooden wedges tightly driven between the two layers of cloth. These

FIG. 53.



COMMON PAN.
(*Plan.*)

wedges, which are driven quite close to each other, must be somewhat shorter than the thickness of the false bottom; thus leaving a space above them which is subsequently covered with a paste of iron cement, that is allowed to set before using the apparatus. About one horse-power is required to work this pan, which will amalgamate from one and a half to two tons of ore in the course of 24 hours.

Varney's Pan.—A drawing of this apparatus to scale is given Plate VII.: fig. 1 is a Vertical Section of this amalgamator; fig. 2, a Plan of the parts beneath pan; fig. 3, Elevation of the amalgamator complete; fig. 4, View of interior of amalgamator; fig. 5, View of one-half the lower dies with wood in slots; fig. 6, View of under side of one-half of muller with shoes attached; figs. 7 and 8, Stand for gear on vertical shaft; and fig. 9, Pillow-block for the driving shaft.

The body of the amalgamator consists of a pan or tub A, figs. 1 and 3, with cover B, through which is an opening for the introduction of the pulp to be ground and amalgamated. The pan is supported on suitable framework, shown in fig. 2. From the centre of the pan and extending from its bottom, to which it is cast, some distance above the cover stands the vertical tube D, through the interior of which is a hole passing vertically through the pan, in order that the shaft C may work through it. On the bottom of the pan, and secured to it by bolts *e*, is fixed the lower muller *a*, consisting of a circular iron plate having a round hole *d* in its centre, considerably larger than the base of the tube D. This die may, if desired, be made in sections.

That portion of the hole through the muller not occupied by the tube D, is so filled with wood as to present a plain surface from the tube to the circumference of the muller. The diameter of this muller is somewhat less than that of the interior of the pan, by which means a space *a'* is left to be filled with quicksilver. Above the lower muller is the upper one *b*, of like general form and size, having twelve shoes *c*, the form and relative positions of which will be understood by supposing a plate of the diameter and thickness of the lower muller attached to the under side of the upper one, and sawn into twelve equal parts on lines drawn from the circumference of the plate to the outside of the tube D. The saw must also be supposed to be held inclined at an angle of about forty-five degrees, thus forming radial grooves from the inner to the outer opening.

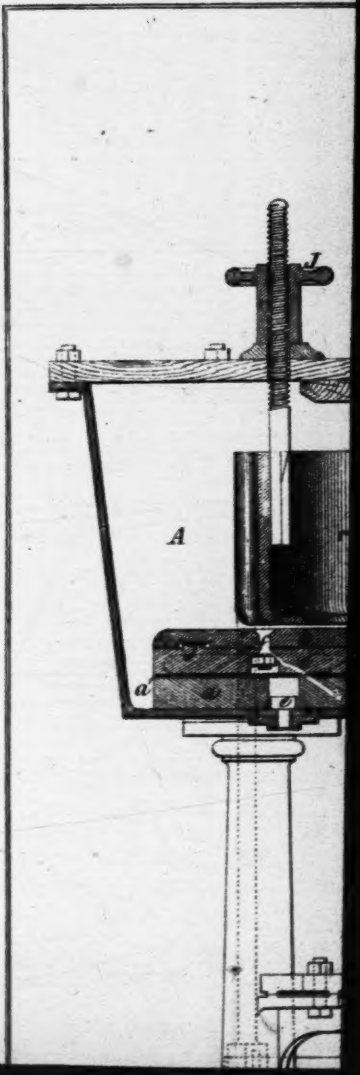
Each shoe is fastened to the muller by a bolt, or a wrought iron rivet, cast into the shoe and riveted into a counter-sink on the upper side of the muller, as shown at *f*, fig. 1; the bosses and recesses *j*, keep the die in its place. In the lower muller are radial slots, similar to those in the upper one. These slots may be either inclined laterally or be made vertical. The slots in the lower muller are filled with wood, so as to grind on its end, in order that it may be kept slightly worn, in advance of the wear of the die; thus furnishing a cavity for the admission of pulp between the surfaces, by which the grinding capacity of the machine is greatly increased.

Over and around the tube D, but not in contact with it, is placed the larger tube E, exactly perpendicular to the lower face of the upper muller, and having around its lower extremity the flange V, upon which rests the ring *h*, which is cast with, and forms a part of, the upper muller. This is connected with the muller by means of six curved arms *i*, two pairs of which are much nearer together than the others, and the space between them is filled by a projection from the periphery of the flange V, for the purpose of carrying with it the upper muller when the flange makes a revolution. To the shaft C is fastened the large tube E by the feather *k*, and set screws *l* in the hub G. The shaft C passes through a Babbet metal bearing at *m*, and through the boss F of the driving wheel, in which is a feather sliding vertically in the shaft. The shaft is stepped, by the ordinary method, into the vertical sliding box H, which is itself held in the laterally adjustable box O.

The step box rests upon an iron bar, one end of which is supported by a screw bolt *w*, fig. 4, and the other is held by a bolt and hand wheel *x*, figs. 3 and 4, by which it can be either raised or lowered; raising or lowering the upper muller at the same time.

Within the body of the pan are suspended three curved plates *r*, figs. 1 and 4, extending from near the surface of the upper muller upwards, and stretching in length from the inner side of the pan around to a point near the outside of the large boss, opposite that from which they started.

The lower edges of the curved plates are bent inwards, as shown at *s*, fig. 4, forming flanges. The inner ends of the curved plates are secured rigidly to the ring *q*, of sufficient diameter to surround and clear the tube E; the whole being suspended by a rod attached to each plate, passing through the cover and hand wheels J, by which it may be adjusted. The outer ends of the curved plates slide vertically in grooves in the projections *t*, cast upon the inner side of the pan. The operation of this apparatus is as follows: The space *a′* about the periphery of the lower muller is filled with quicksilver, and the pan


J
A
a
o

VARNEY'S PATENTED

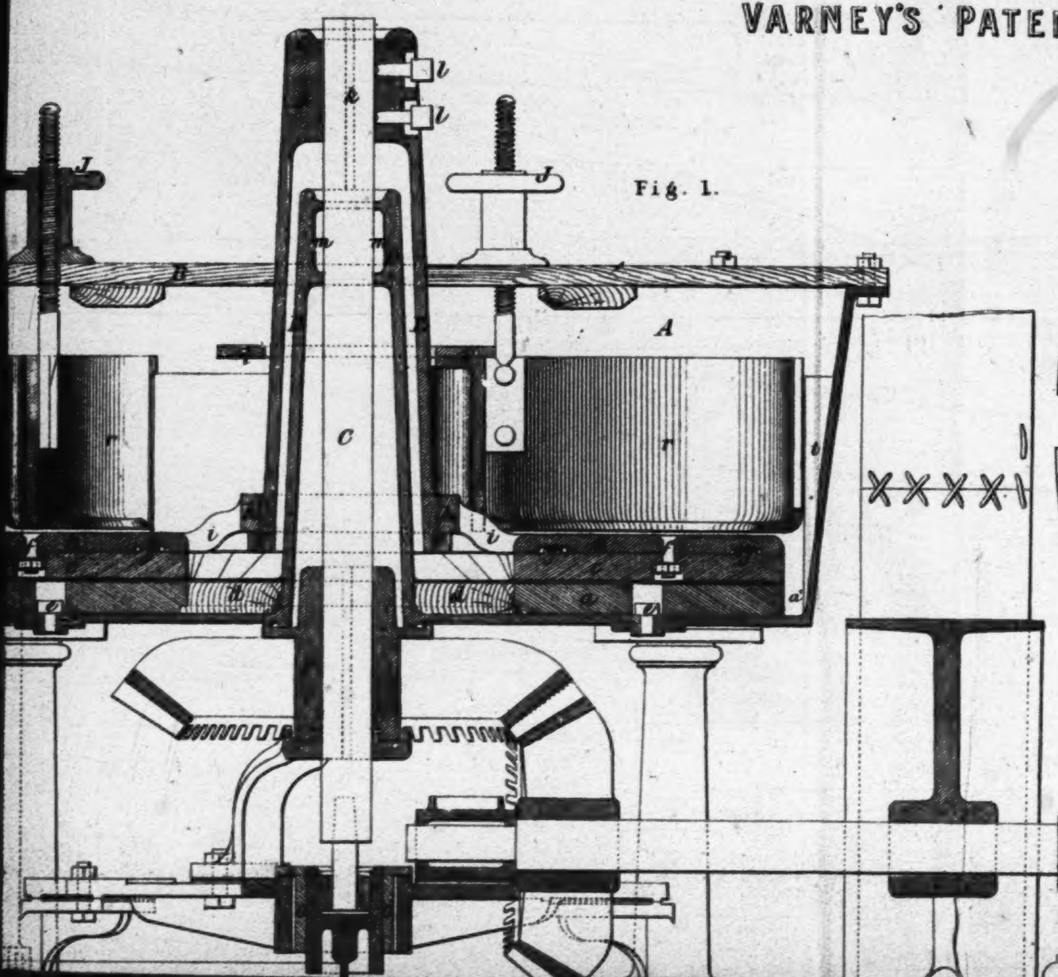


Fig. 1.

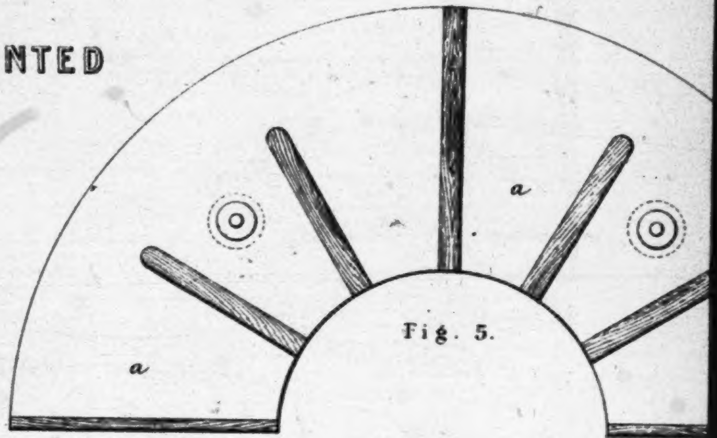


Fig. 5.

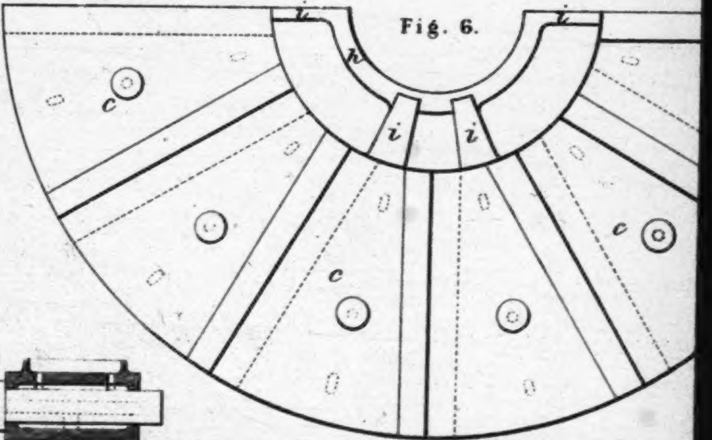


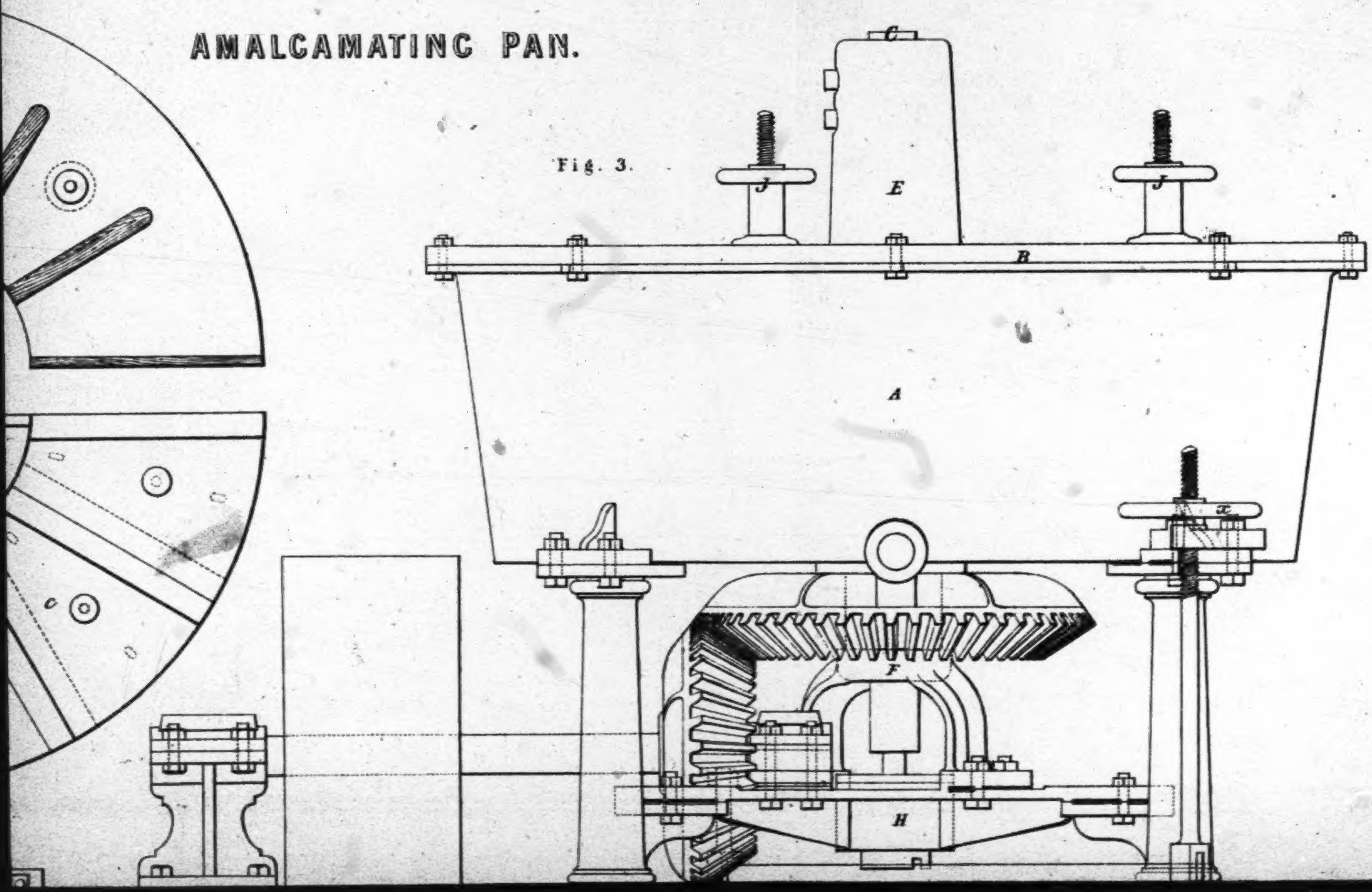
Fig. 6.

Fig. 7.

PLATE VII.

AMALGAMATING PAN.

Fig. 3.



Inches 12

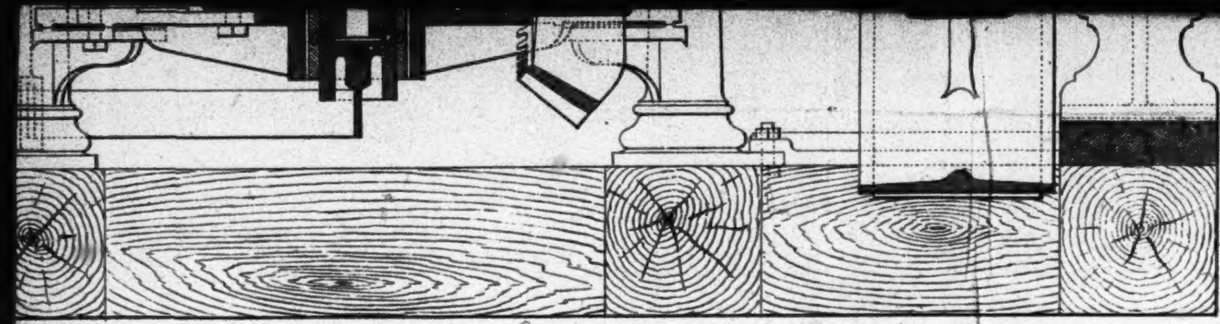
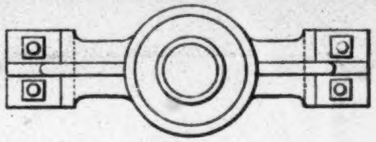


Fig. 7.



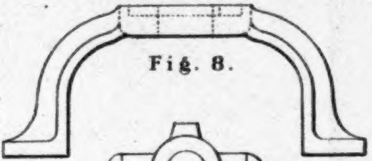


Fig. 8.

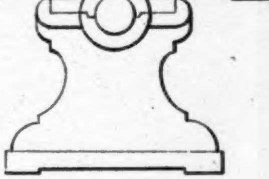
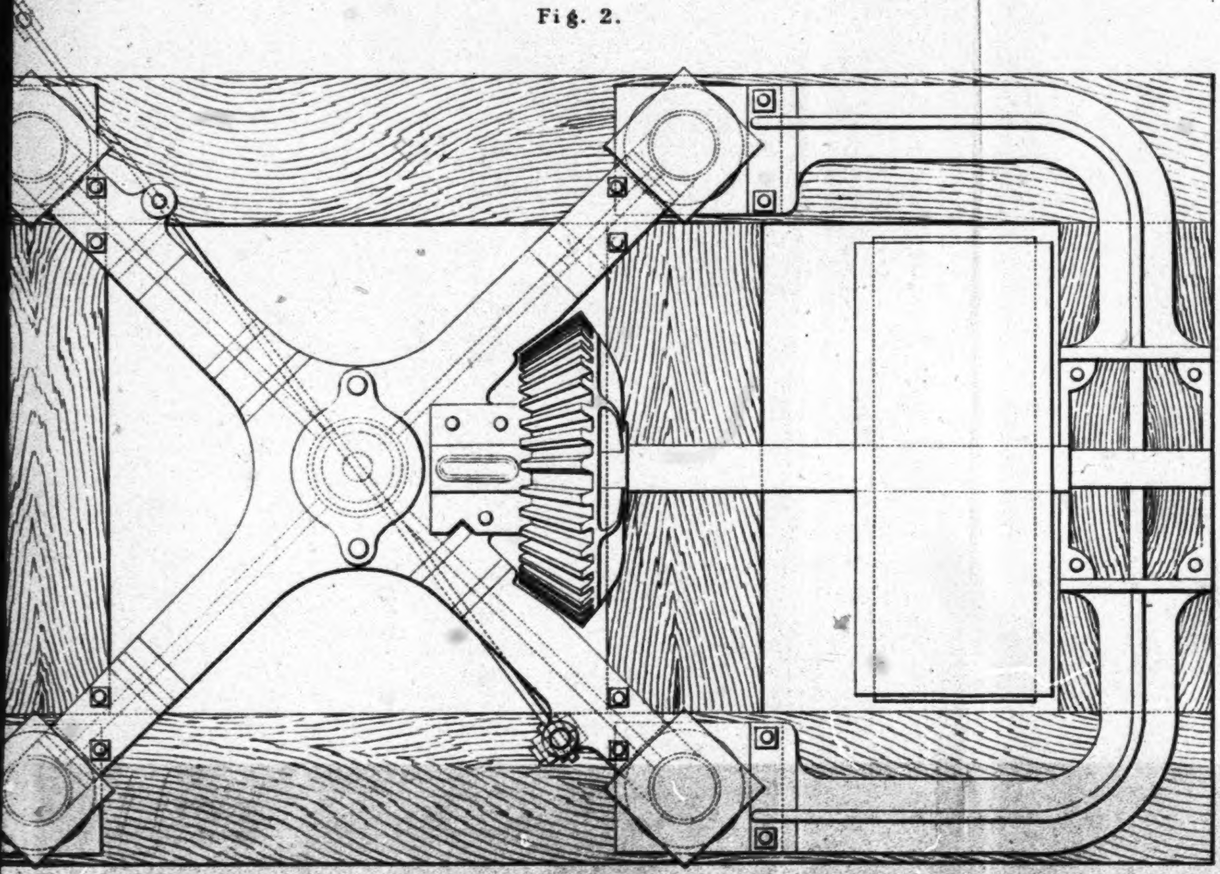


Fig. 9.

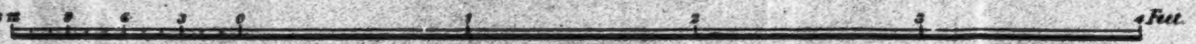
Fig. 2.



REFERENCE.

A.	*Pan or tub.*
B.	*Cover.*
C.	*Vertical shaft.*
D.	*Central tube.*
E.	*Exterior tubes.*
F.	*Boss of driving wheel.*
G.	*Hub of outer tube.*
H.	*Sliding bearing.*
J.	*Hand wheels.*
O.	*Adjustable box.*
V.	*Flange of outer tube.*
a.	*Lower muller.*
a'.	*Space for mercury.*
b.	*Upper muller.*
c.	*Shoes of „*
d.	*Hole in centre of lower muller.*
e.	*Bolts securing „*
f.	*Bolts securing shoe to upper „*
h.	*Ring on upper muller*
i.	*Curved arms of „*
k.	*Sliding key.*
j.	*Lugs to keep shoes in place.*
l.	*Set-screws.*
m.	*Babbit Metal bearing.*
q.	*Ring supporting curved plates.*
r.	*Curved plates.*
s.	*Flanges of curved plates.*
t.	*Brackets supporting „*
w.	*Fulcrum of lever.*
x.	*Hand wheel for lifting muller.*

SCALE OF FEET.



E. & F. N. Spon. 48, Charing Cross, London.

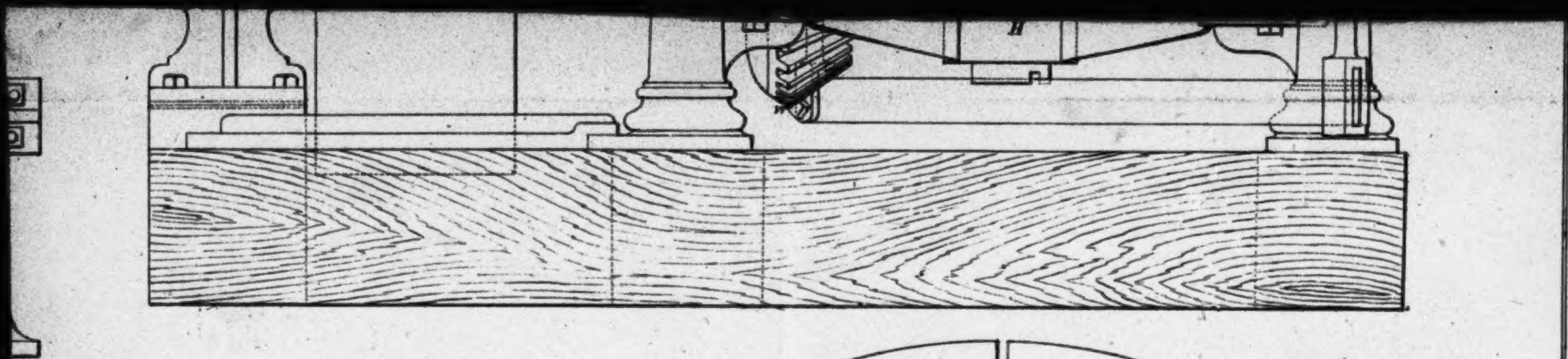
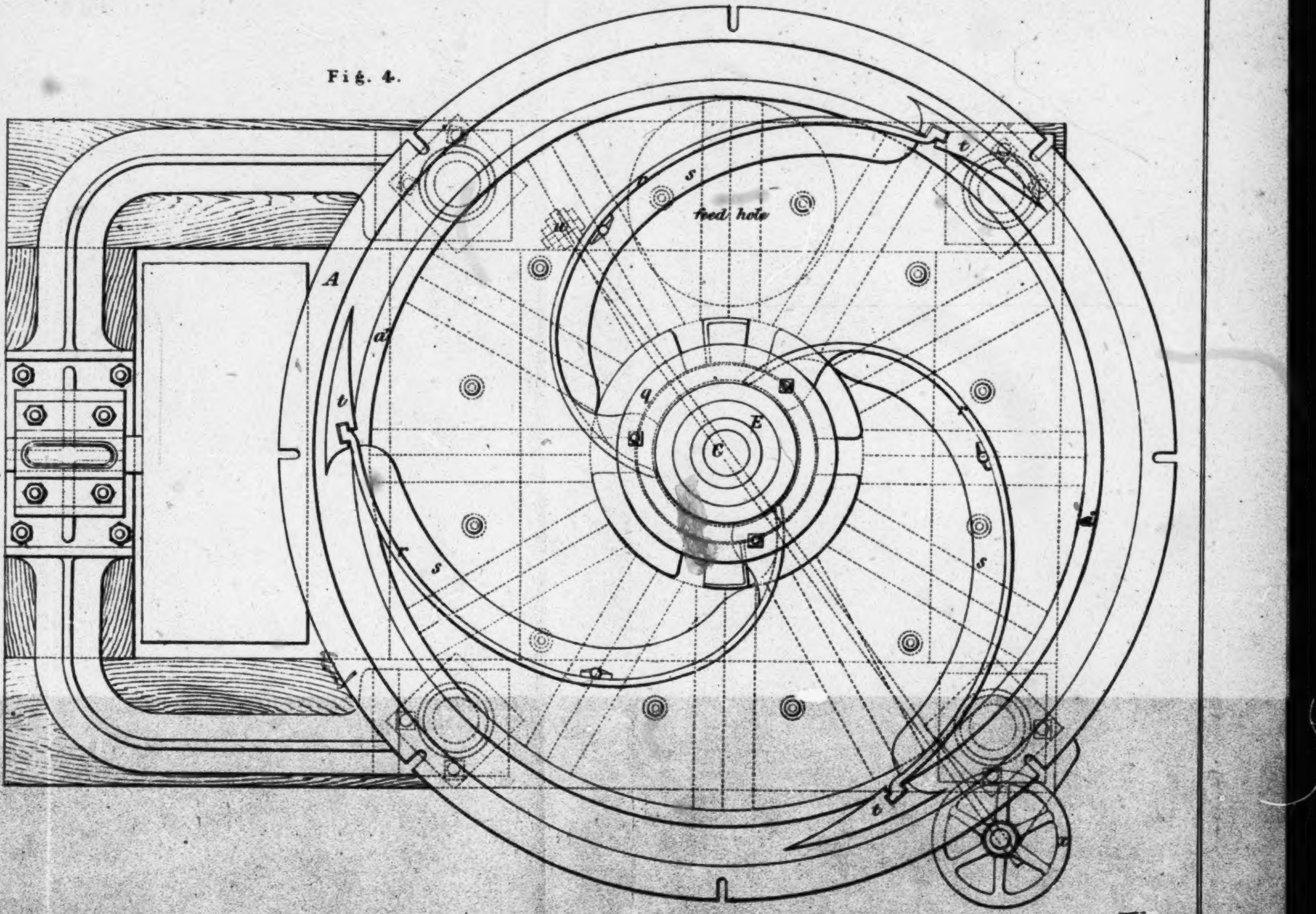


Fig. 4.



Newbery & Alexander, Lith. 43 Castle St. Holborn.

ODD FE · SAN FRANCISCO, CAL.

nearly filled with pulp, of the proper consistency to flow easily; the shaft C is now made to revolve at a proper speed, from sixty to eighty revolutions per minute, by which the upper muller is rotated. The pulp between the mullers, by means of the centrifugal force developed, is made to pass out through the radial channels between the dies, as well as between the grinding surfaces of the upper and lower mullers; also into and over the quicksilver, thereby causing amalgamation.

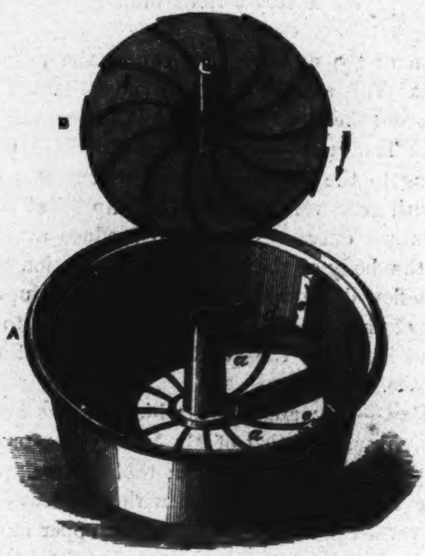
The outward motion of the pulp has the effect of keeping the quicksilver entirely away from the grinding surface, thereby obviating what has often proved a very serious difficulty—viz. the grinding of the mercury.

The rotation of the upper muller causes the pulp in the pan to revolve with it. This current is met by the cuneiform projections and curved plates, and thereby turned towards the central opening in the upper muller. The radial slots between the shoes, running from the central opening to the outward one, allow currents of considerable size to pass with great velocity; and the pulp filling these slots, being continually thrown outwardly, tends to produce a vacuum. By this the pulp in the body of the pan is set in motion, causing a rapid and abundant flow downwards at the centre, and upwards along the inner surface of the pan. The pulp is thus made to circulate, until complete pulverisation of the quartz and amalgamation of the metals have taken place.

Wheeler's Pan.—This apparatus, which in size and certain other respects closely resembles that of Mr. Varney, is represented Fig. 54; A being the pan, with the dies *a*, in their several places; whilst B represents the rotating muller, fitted with its shoes *b*, removed from the pan, and turned bottom upwards. The upper muller is, like that of the pan last described, driven by means of a hollow cone, which passes over the central pillar, and is connected with the vertical shaft by means of a sliding key.

As in the case of the other pans, the distance between the mullers is regulated by a screw, fitted with a hand wheel. The shoes *b* are secured to the upper muller, either by bolts and nuts, or more frequently by projections passing through inclined oblong holes in the rotating plate, to which they are firmly secured by means of wooden wedges. The dies *a* are laid on the bottom of the pan, and kept in their places by the ring *c* in the centre, and on the sides by the inclined ledges *d*, under which their ends are wedged. The dies, like the shoes, are one inch thick, and bevelled on the edges in the same direction; so that, when put together, grooves are formed between them, as shown in the drawing. On the upper side of the outer edge of the muller are inclined ledges, which, in connexion with those, *d*, cast on the pan, create an upward current in the pulp; whilst guide plates, somewhat similar to those of the Varney Pan, which slide into grooves at *e*, convey it towards the centre. This pan stands on a cast iron framing, and is driven by mitre wheels from beneath.

FIG. 54.



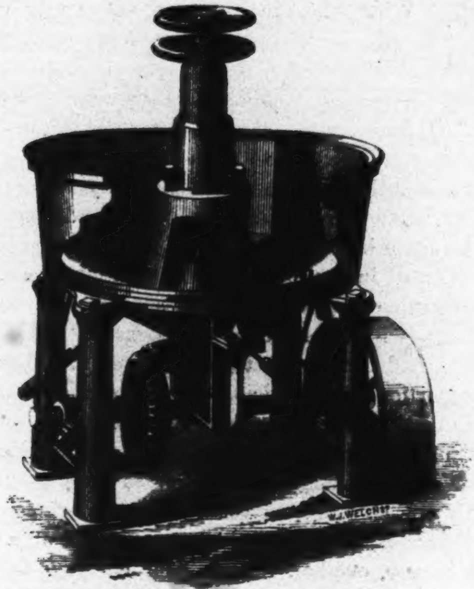
WHEELER'S PAN.

From the dies and bottom not being cast perfectly true, the grinding surfaces are often, at first, a little uneven, and consequently the grinding planes should not at once be brought into too close contact.

The runner of these pans requires to be lifted at least once a week, for the purpose of removing the amalgam which accumulates around the central pillar, and thus prevents the pulp from passing freely between the grinding surfaces. This pan, like that previously described, is generally made four feet in diameter at bottom, and requires from two and a half to three horse-power to work it efficiently. It usually makes about sixty revolutions per minute.

Hepburn and Peterson's Pan.—This pan differs mainly from the foregoing, as will be seen by the annexed illustration, Fig. 55, in the shape of the bottom, which is inclined towards the centre, or shaped like an inverted cone. The shoes are bolted to this cone, and the corresponding dies fastened to the bottom. When the pulp is thrown into this apparatus, and the mullers set in motion, that portion of it which finds its way between the grinding surfaces is thrown towards the circumference, from whence it again descends by gravitation to the centre, and passes between the mullers. A constant and active circulation is thus established without the aid of curves or wings; which have sometimes been found an impediment in starting similar machines, after the sand has become packed from stopping. Under all ordinary circumstances, however, this or any of the other well-constructed amalgamating pans may be readily started, without either

Fig. 55.



Hepburn and Peterson's Pan.

removing or thinning down the pulp, by simply elevating the muller, by means of the screw and movable nut with which they are now generally provided. The charge for this pan is about 1,400 lbs. and the time required for working it is from two to four hours, in accordance with the fineness of its state of division and other characteristics. When the ore has been sufficiently reduced and amalgamated, the pulp is, after dilution with water, discharged into another pan, called a *separator*, and the amalgamating pan immediately re-charged, without stopping the machine. After the pulp has been run off into the separator, it is further thinned down with water to such a consistency as will allow the mercury and amalgam to settle, whilst it still retains sufficient plasticity to hold the coarser particles of ore in suspension in water. The condition of the pulp is readily ascertained by placing the hand in it during the process of separation. If it be in a proper state of dilution, the mercury and amalgam will gradually precipitate, and at the same time no perceptible difference will be felt in the consistency of the pulp situated near the bottom and that at the top of the vessel. When, however, too large a proportion of water has been added, the coarser particles will be felt to distinctly separate from the slime, and strike against the hand when placed near the bottom of the separator. In working these, or other similar machines, the charges are generally so regulated that only one charge, from the two pans, working in conjunction with a separator, may be ready to operate on at a time; thus taking, in each case, one-half the time for effecting the separation that is consumed in reducing and amalgamating. Some amalgamators, however, prefer to occupy the same length of time in effecting the separation that is required for amalgamation; and in this case the separators require to be of larger dimensions, or to be more numerous, since both pans are run off together.

The Hepburn and Peterson Pan is much employed in the reduction establishments of the Pacific Coast, and, in addition to being an excellent amalgamator, is also a good grinder; but it has the disadvantage of requiring the expenditure of from four to five horse-power for its efficient working. The charge of the Wheeler Pan is not only less than that of the Hepburn and Peterson Pan, but its grinding power is also less considerable. This pan usually makes between fifty and sixty revolutions per minute.

Separators.—These differ more or less in their details, but generally consist of a large wooden tub, having a considerably greater diameter than the pans, and provided with a cast iron bottom. Arms and mullers are attached to a shaft working through the centre of this bottom, as explained when describing the construction of the common pan. In this case, however, the mullers are sometimes made of mere blocks of hard wood, the object being simply the agitation of the pulp for the purpose of concentrating at the bottom the quicksilver and amalgam run off from the pans. The mullers in the separator are only allowed to make from ten to twelve revolutions per minute.

Working in Pans.—The reduction process simply consists in treating the pulverised ore in cast iron pans in such a way as to cause the amalgamation of the gold and silver it contains. The same principles are to a great extent involved in this process as in the Mexican patio; but, by intelligent modifications of the treatment, as much is accomplished by the former in a few hours as can be effected by the latter in as many weeks. Although the improvement on the old Mexican process already made has been very great, much yet remains to be accomplished. In order to the most complete and perfect separation of the metallic from the earthy constituents of the ores, it is first necessary that they should be reduced to an impalpable powder by grinding. This is but imperfectly done by the stamping mill, and much is left to be accomplished by the pans. The friction of the pan should not, however, be more than sufficient to insure the perfect mixture of the ingredients which it contains, in such a way as to promote a rapid series of chemical changes, resulting in the decomposition of the constituents of the ore, and the combination of the precious metals with mercury, at the least possible cost of power and material.

With this view, from twelve hundred and fifty to fifteen hundred pounds of ore, from the tanks of the stamping mill, are put into the Varney or Wheeler Pan, and the grinding muller gradually lowered until the whole mass has become reduced to an impalpable powder. This is usually accomplished in about an hour. Loose steam is then turned on until the temperature has been raised to 200° Fahr.; but care is at the same time taken not to render the pulp too liquid by the accumulation of condensed water. The muller having been slightly raised, to prevent too great an amount of friction between it and the dies, quicksilver is gradually added in the form of a fine shower, by pressing it slowly through a canvas bag. The quantity of quicksilver used varies, according to the richness of the ores, to from ten to fifteen per cent. of the weight of mineral operated on. Amalgamation is further promoted by the addition to the pulp, immediately after introducing the quicksilver, of sulphate of copper and a small quantity of sulphuric acid.

For second class ores, two pounds of sulphate of copper, in solution with an equal quantity of sulphuric acid, and three pounds of salt, may be advantageously employed.* An endless variety of substances has been used for this purpose, since at one period every amalgamator prescribed some new specific for avoiding imperfect amalgamation; but those above enumerated have alone maintained a permanent character for efficiency.† Much, however, yet remains to be accomplished by a more careful study of the modifications rendered necessary by the varying constitution of the ores operated on. The running of the pan is continued for three hours and a half, the heat being maintained at from 180° to 200° Fahr. At the expiration of this time, water is run into the pulp to render it sufficiently liquid to flow off, through a valve in the bottom of the pan, into the agitator or separator. The pan is then roughly washed down, and, with as little delay as possible, recharged with ore, as before. A Varney Pan four feet in diameter is capable of working six charges in twenty-four hours, or from 7,500 to 9,000 lbs. of ore. Once or twice a week, or at the finishing of any particular parcel of ore, the muller is taken out, the shoes and dies removed, and all the amalgam adhering to the working parts, or deposited in the crevices, carefully scraped off. A notable accession to the product is thus obtained, especially as, at the same time, the mortar bed of the stamps is likewise cleaned out.

In the separator the pulp is mixed with a large quantity of water, and a regular steady supply kept up, so as to carry off the lighter particles of earthy matter, at first from holes in the upper part of the pan; but as the separation proceeds, the discharging point is gradually lowered, until eventually nothing but the heavier pyrites and liquid amalgam is left. The amalgam is drawn off from the bottom, and the pyrites then scooped out, and, after being further washed in another separating pan, to remove the last traces of amalgam, it is reserved for final treatment by calcination and reduction in barrels. The amalgam is now carefully washed in clean water, dried with flannel, and finally removed to the amalgam room, where it is strained through thick conical bags of canvas, twelve inches in diameter at the larger end and two feet in length.* After the bags have drained for some time, they are beaten with a round stick to cause a further quantity of the mercury to run off. The hard dry amalgam is finally removed from the bags, and weighed into store. The mercury run off from the bags is technically known as "charged quicksilver," and, after being mixed with retorted mercury, is returned to the pan room for further use. Charged quicksilver is preferred to the pure metal, as with it amalgamation is found to proceed more rapidly.

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* Dr. Oxland, who has had considerable experience in the treatment of silver ores by the pan process, informs us that he has obtained as good results with sulphate of copper alone, as with the addition of salt and sulphuric acid.

† It is scarcely necessary to notice the various absurd ingredients which have occasionally been employed in Nevada with a view of facilitating amalgamation, such, for instance, as tobacco juice, decoction of sage bush, and of oak bark; since any success which may have attended their use must manifestly be attributed, rather to a fortuitous combination of circumstances, than to any direct action of the materials themselves.

Retorting.—The amalgam is finally handed over to the assay office belonging to the mill, and the separation of mercury is effected by exposing the amalgam to a red heat in a cylindrical cast iron retort, about twelve inches in diameter and three feet long, mounted on an arch of firebrick, and placed within another arch, from the crown of which the smoke is carried off to the chimney. The retort is fitted with a stout cover, carefully adjusted like the stopper of a coal-gas retort. From the upper part of the end, a two-inch iron pipe carries off the volatile matters. This is so fitted to the downcast pipe, four feet in length, that, by T-pieces and stoppers, every facility is afforded for cleaning out the pipes. The downcast pipe is so fitted within another pipe, 3½ to 4 inches in diameter, as to constitute a Liebig's condenser, into the bottom of which cold water is supplied; the heated water flowing off from the top. The downcast pipe opens into a small bottomless chamber, immersed sufficiently low in a tank of water to keep it air-tight, but in such a manner as to prevent accidents from the absorption of water into the heated retort.

This retort is provided with several cast iron semicircular trays, which slide easily in and out; these are divided into two parts by a transverse partition. Before the weighed charge of amalgam is put into the tray, it is coated with milk of lime, or a thin wash of clay, and not unfrequently a sheet of paper is also placed over the bottom. By these precautions the retorted amalgam is prevented from adhering to the iron, and much trouble avoided. The charge having been placed in the retort, the cover is carefully luted with a mixture of clay and wood ashes, made up into a thin paste. The fire is then lighted, and the heat slowly and steadily raised, until the retort is of a bright red colour, and is so maintained until the mercury ceases to distil over. The retort is now allowed to cool gradually down, and when cold the retorted silver is withdrawn and weighed, as is also the mercury obtained, as a precaution against any possible loss of quicksilver from hidden leaks in the retort. The retorted amalgam is broken up, melted in plumbago crucibles, and cast into bars or ingots of "bullion" of from one thousand to fifteen hundred ounces each. These are assayed and valued, the value being marked on the bars, which are then ready for the market. The quality or "fineness" is marked in thousandths, thus—gold 24, silver 841, making together 865 thousandths; leaving 135 parts in a thousand, which principally consist of copper; but no notice is taken of this, as it is of no money value in the sale of the bar.

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* The washing of the amalgam is generally effected in a vessel called a cleaning up pan, provided with four arms furnished with shoes like those represented, Figs. 52 and 53. In some cases, however, the matters escaping from the separators are now run off over blankets on which a certain proportion of the undecomposed sulphides is caught, and this is subsequently re-treated either by barrel amalgamation or otherwise.

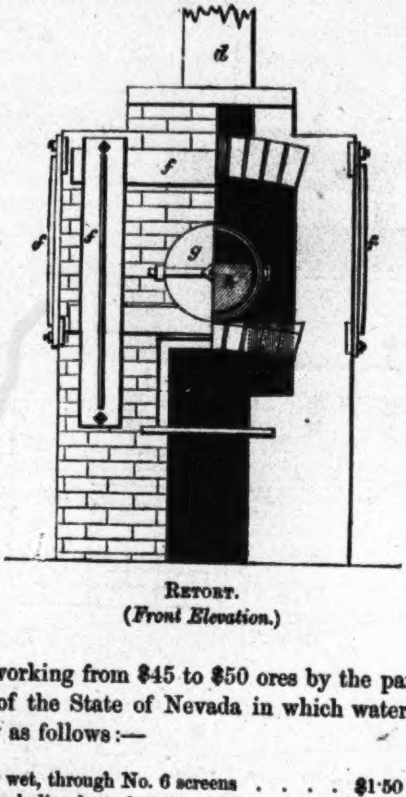
The retort employed at the mills near Virginia for the distillation of silver amalgam is represented, Figs. 56 and 57, of which the first is partially in section, and the second is a longitudinal section.

The ash-pit A is beneath the fireplace B, which communicates, by means of flues *a*, with a chamber *b*, enclosing the cast iron retort C, from which the products of combustion are conveyed by the flues 1, 2, 3, through the arched cavity *c*, to the chimney *d*.

By dampers covering these flues the draught may be controlled so as to heat the retort according to the requirements of the case. The pipe D carries the vaporised mercury to the vertical pipe E, in which it is condensed, by the action of a stream of cold water passing upward from the bottom through the Liebig's condenser F. The condensed mercury collects in the reservoir G, from which it is drawn off into bottles through a bent tube at the bottom. Any vapours escaping from the retort door are conveyed into the flues by the hood *e*, of sheet iron. The arrangement of the cover of the retort is shown at *g*, and a portion of the semi-cylindrical tray, used for charging the retort, at *h*; the position of the iron plates and braces for binding the brickwork is represented by the letters *f*.

The best results obtained by the pan process rarely amount to 75 per cent. of the assay value of the ore; the average will scarcely exceed 65 per cent. The barrel process, being more expensive, cannot be employed in Nevada except on ores of the first class. The tailings from the pan process, after having been exposed to the action of the atmosphere for a few months, may sometimes be again advantageously worked over, thus increasing the total produce to about 85 per cent.; but it is only under favourable circumstances that this can be done.

FIG. 56.



RETORT.
(*Front Elevation.*)

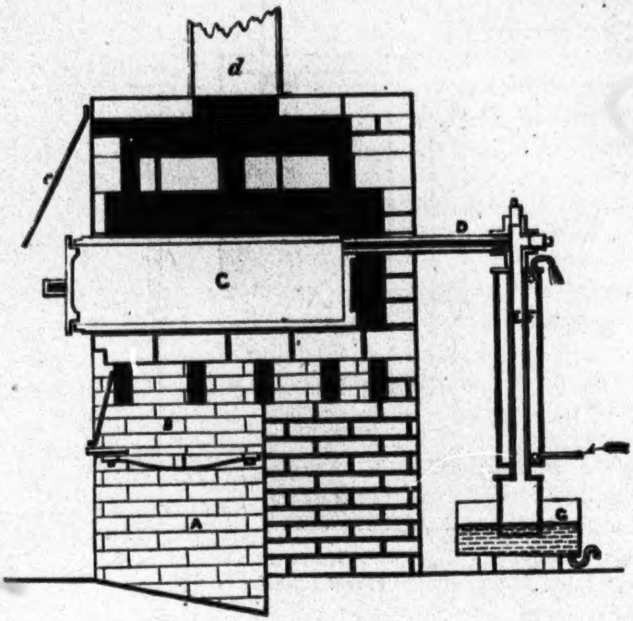
The cost of working from $45 to $50 ores by the pan process is, in those portions of the State of Nevada in which water-power can be obtained, nearly as follows:—

Stamping wet, through No. 6 screens	$1·50 per ton.
Milling, including loss of mercury, &c.	5·00 „ „
Total cost, including wear and tear . .	$6·50 „ „

The loss of mercury amounts to from 1¼ to 1½ lb. for each ton of ore containing silver to the amount of from $35 to $50 per ton.

Arrangement of Reduction Works.—The usual arrangement of crushing and amalgamation works for the treatment of silver ores by the pan process, where water-power is available, will be understood by reference to Plate VIII., which represents a transverse sectional elevation of an establishment in Nevada, in which Wheeler's Pans are employed. The water-wheel A communicates its motion, by means of toothed segments around its periphery, to a pinion shaft on which is the drum B, which, in its turn, transmits the power, by means of broad composition belts, to the whole of the machinery employed in the establishment. The dimensions of the second pulley C, and that on the shaft D, are so calculated as to cause each stamper to be raised from 70 to 80 times per minute. The stops *d* are employed for supporting the stampers when not in action, and are for that purpose

FIG. 57.



RETORT.
(*Longitudinal Section.*)

slipped under the bosses or tappets on the iron stems, when raised to the full extent of their course by the cams keyed on the shaft set in motion by the pulley D. The tightening pulleys E are for the purpose of keeping the different straps in a proper state of tension. The stampers F discharge the pulverised ore, through the grates or screens *f*, into the spouts *g*, which conduct it to the receiving tanks G,G′. The hand wheel H is employed for opening and closing the valve *h*, by which the supply of water to the wheel can be either regulated or cut off. From the amalgamating pan I, the *slums* are run off, through the spouts *i*, into the separator K, which is provided with the overflow *k*. The cleaning up pan, L, with the other pans and separators, receives its motion from belts driven by pulleys on the shaft O. The pipe M affords the necessary supply of water, whilst N is the steam pipe by which the heating of the pulp is effected. In the drawing, the pan is represented as having a false bottom, for the purpose of heating the pulp by the introduction of steam, but in practice it is generally now found more convenient to blow loose steam directly into the pulp.

The hand wheel P is for the purpose of regulating the tightening pulley E. This mill has twenty stampers, twelve Wheeler Pans, six separators, and two cleaning up pans.

Amalgamation of Roasted Ores in Pans.—In some of the mining districts of Nevada, and particularly in the neighbourhood of Austin, where the ores consist of various compound sulphides of silver, containing a considerable amount of antimony, the ordinary pan process, as practised in Virginia, cannot be advantageously employed. The ores from this part of the State consequently require roasting before being subjected to amalgamation, and then, when worked in the pans, afford better results than those obtained from the ores of the Comstock vein treated in their raw state. Each battery of five stampers will crush (dry) four tons of ore daily, through a wire gauze screen of 40 holes per linear inch. One thousand pounds of this crushed ore is roasted with 8 per cent. of common salt; the time occupied in the furnace by each charge being, on an average, six hours. Varney's Pans are most commonly employed, and are charged with from 800 to 1,000 lbs. of roasted ore, which occupies five hours in working.

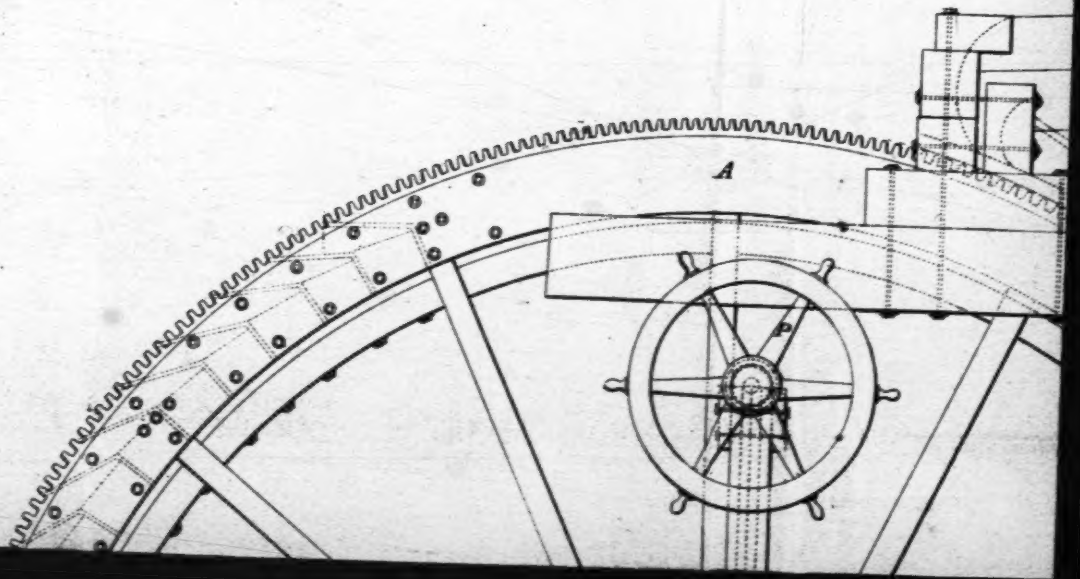
A mill of ten stampers, with all the necessary furnaces, pans, and appliances, will treat eight tons of ore in the course of twenty-four hours, with a total consumption of about ten cords of wood. The following figures, relative to the treatment of roasted silver ores by pan amalgamation, give the results of an experiment made at Virginia city on ores from the Comstock vein, but it is stated that the loss of silver in the neighbourhood of Austin, where the ores contain little or no gold, seldom exceeds 7 per cent. of the assay value.

Results of an Experiment showing Loss of Metal by Amalgamation of Roasted Ore in Pans.

Value of ore in gold	$112·85,	in silver $180·37 . . .	$293·22
„ Bullion	97·43,	„ 162·99 . . .	260·42
„ loss „	13·60 per cent.	9·06 per cent.	11·18 per cent.

Loss of quicksilver greater than in barrels.

ELEVATION
OF
SILVER REDUCTION WORKS,
AUBURN
STATE OF NEVADA.



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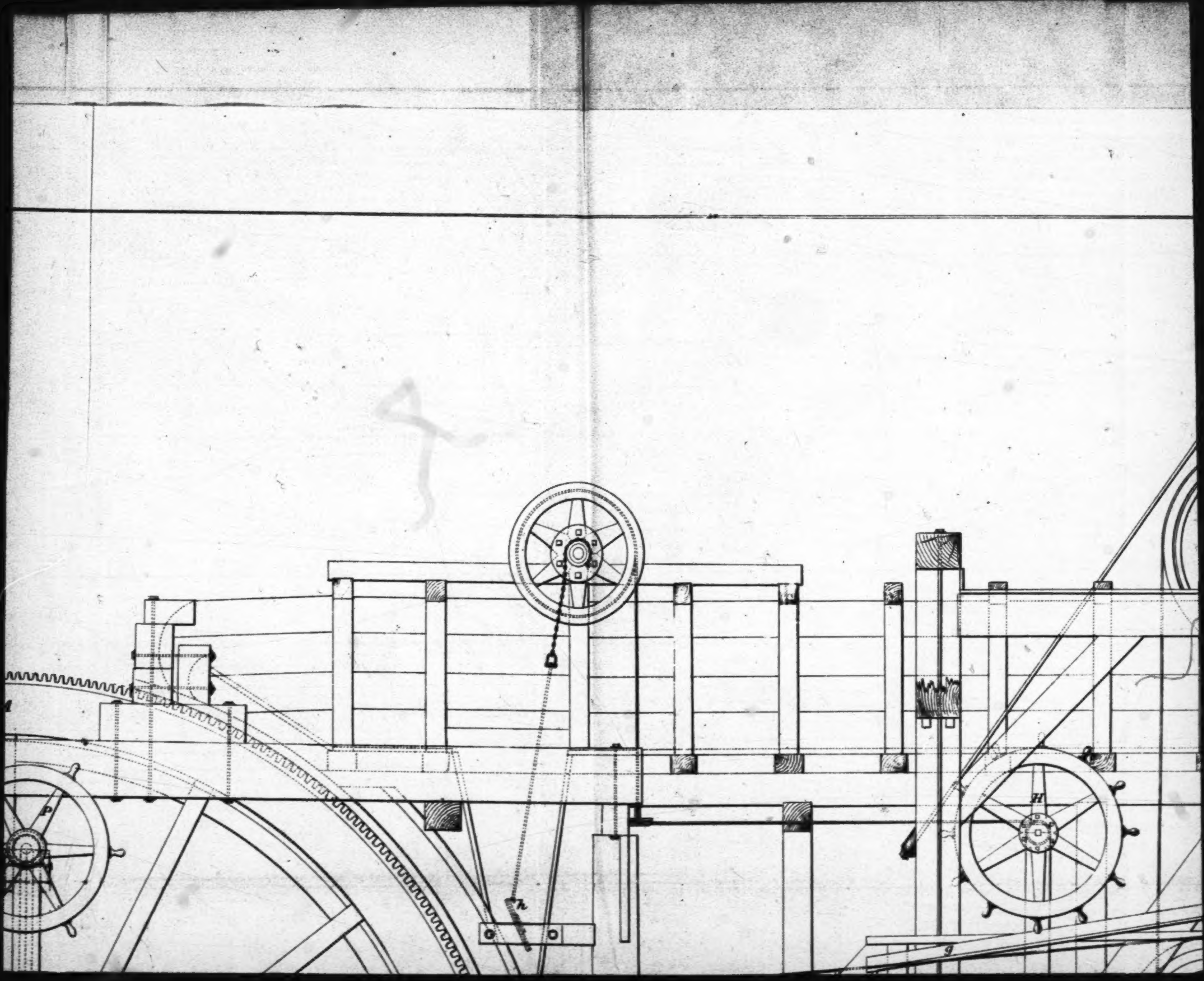

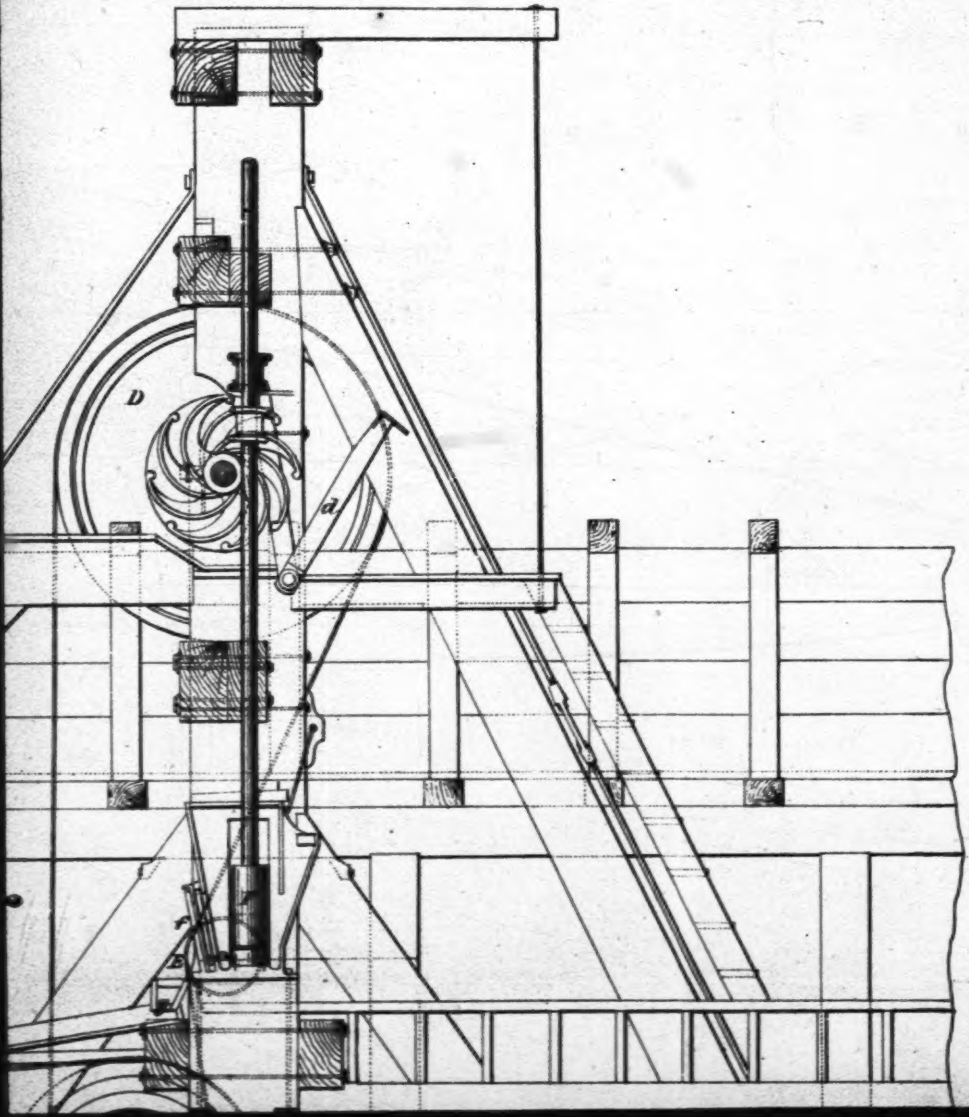
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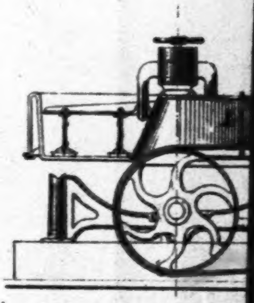

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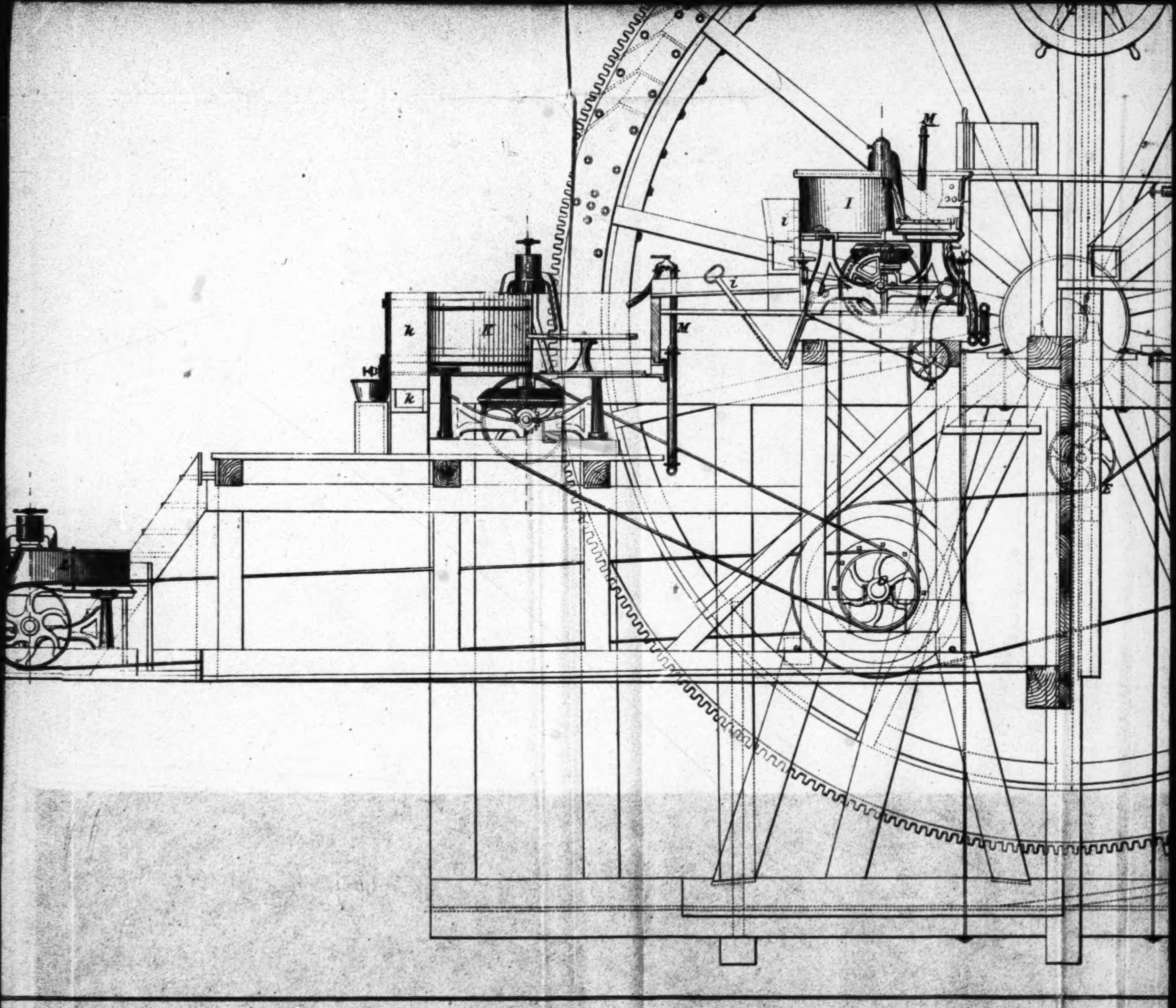
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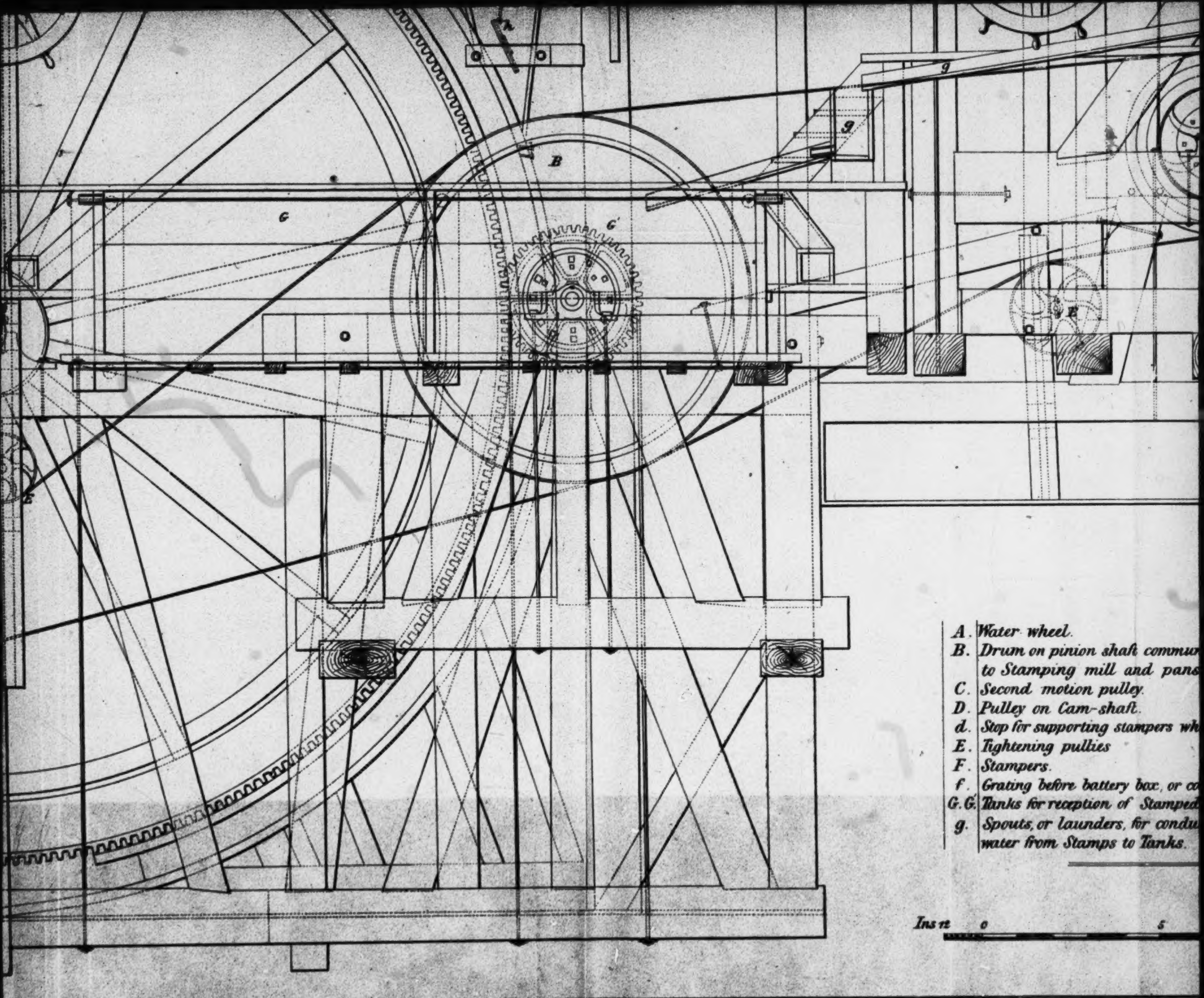
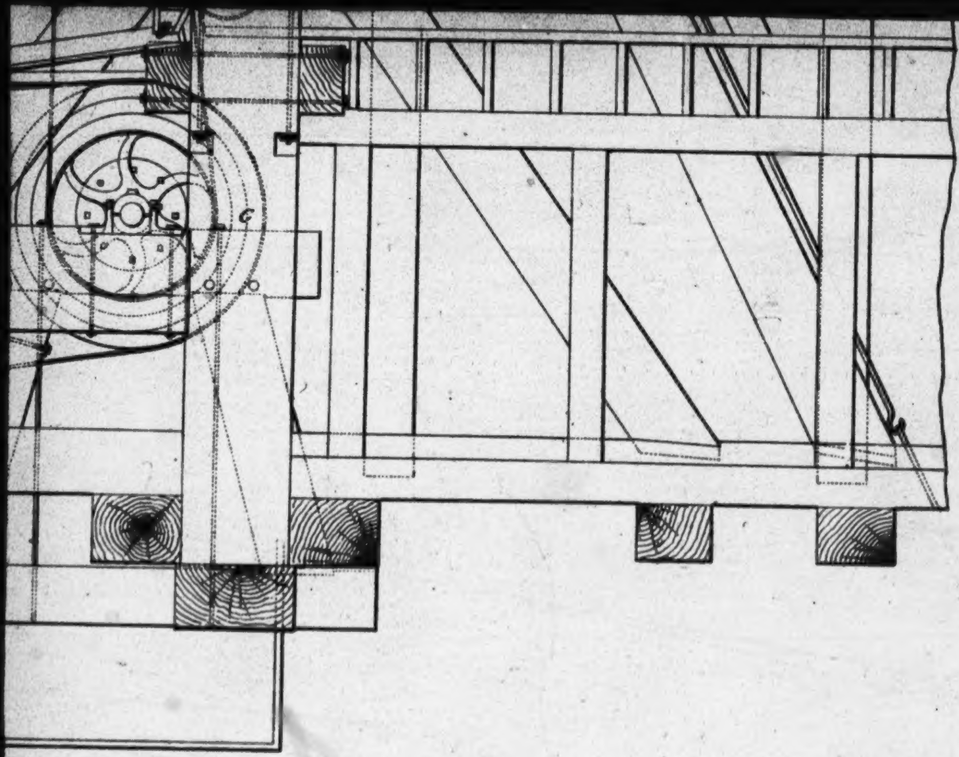


A.	Water wheel.
B.	Drum on pinion shaft commun to Stamping mill and pans
C.	Second motion pulley.
D.	Pulley on Cam-shaft.
d.	Stop for supporting stampers wh
E.	Tightening pullies
F.	Stampers.
f.	Grating before battery box, or co
G.G'.	Tanks for reception of Stamped
g.	Spouts, or launders, for condu water from Stamps to Tanks.

E. & F. N. Spon, 48, Charing Cross, London.



REFERENCE.

communicating motion
d pans.

pers when not in action.

ox, or coffer.
Stamped Ore.
conducting ore and
Tanks.

H. Hand wheel for opening and closing valve h.
I. Amalgamating pan.
i. Spouts for running contents of pan into Separator.
K. Separator.
k Overflow pipe from Separator.
L. Cleaning up pan.
M. Pipes for water supply.
N. Steam Pipe.
O. Shaft with pulleys, giving motion to pans separators, and cleaning up pans.
P. Hand wheel for lifting tightening pulley E.

SCALE

5 10 15 20 Feet

CHAPTER XIX.

TREATMENT OF ARGENTIFEROUS ORES AND PRODUCTS BY SOLUTION AND PRECIPITATION.

AUGUSTIN'S PROCESS—ROASTING WITH SALT—LIXIVIATION AND PRECIPITATION—ZIERVOGEL'S PROCESS—ROASTING—SOLUTION OF SULPHATE OF SILVER—PRECIPITATION OF SILVER BY COPPER—VON PATERA'S PROCESS—ROASTING WITH COMMON SALT—SOLUTION OF CHLORIDE OF SILVER IN HYPOSULPHITE OF SODA—PRECIPITATION BY SULPHIDE OF SODIUM—TREATMENT OF SULPHIDE OF SILVER.

THE different processes by which silver is obtained by the *wet way* from the various ores and metallurgical products containing that metal are all of recent invention, and belong to the latest period in the history of metallurgy. These methods have now, in many cases, supplanted the older processes of liquation and amalgamation, and may be often advantageously adopted for the treatment of argentiferous compounds; particularly when the amount of lead present is small, and the proportion of copper large. They all possess the advantage, over amalgamation, of entailing no loss of mercury, and in the promptness and completeness with which the metal may be extracted. They are also, from requiring less time and fuel, and in being attended with a smaller loss of the various metals contained in the substance under treatment, less expensive than fusion with lead ores. The several wet processes are likewise more expeditious, and require a smaller amount of fuel than the old method of liquation; besides which, they effect a much more complete separation of the copper and silver.

AUGUSTIN'S PROCESS.—This method of treating argentiferous compounds was first introduced in 1849 by an officer of the Mansfeld Copper Works, at the Gottesbelohnung Works, near Eisleben, in Prussian Saxony, where, after being retained for a short time, it was superseded by the cheaper and still more simple process of Ziervogel. Augustin's method of extracting silver from its ores is dependent on the following circumstances:—

1st. That the silver contained in ores of that metal may be converted into chloride of silver by roasting them with a proper admixture of common salt.

2nd. That a solution of common salt will dissolve chloride of silver in quantities depending on its temperature and state of concentration.*

3rd. That the silver contained in an argentiferous solution of chloride of sodium is precipitated in the metallic state by copper. At Freiberg this system is employed for the extraction of silver from a regulus or copper matt, containing about 70 per cent. of copper and 0·0042 of silver, besides a certain amount of iron, antimony, and arsenic, with other impurities. In order that this matt may be properly roasted, it is first ground to a very fine powder and bolted through sieves of wire gauze.

Roasting.—The furnace employed for this operation is of the ordinary reverberatory description, the fuel employed being pit coal, and the weight of the charge treated four hundred pounds. One workman attends to each furnace, and the fire, which at first is kept low, is gradually raised, care being at the same time taken to keep the powdered matt constantly stirred, in order to prevent caking, and that every portion may be equally exposed to the full temperature of the hearth. At the termination of eight hours the operation begins to approach completion; the bright glow, indicative of the presence of sulphur, disappears, and sulphurous acid is no longer evolved. The charge is now withdrawn from the furnace, and, after being allowed to cool, is ground between burr stones, passed through a fine bolt, and subjected to chlorination.

Roasting with Salt.—The roasted matt, in which, at this stage of the operation, the copper and iron chiefly exist in the state of oxides, whilst the silver has either been converted into a sulphate or reduced to the metallic state, is now roasted in charges of three hundred pounds weight, in furnaces similar to those employed for the first operation. After being for a short time exposed to a low temperature, common salt, to the amount of 5 per cent. of the weight of the charge, is introduced, and the roasting continued, with the usual amount of stirring, for about three hours. By this means the decomposition of the chloride of sodium is effected through the agency of the sulphuric acid of the metallic sulphates; and, by the combination of chlorine with the silver present, nearly the whole of that metal becomes converted into chloride. The charge is now withdrawn from the furnace, and is ready for the process of lixiviation.

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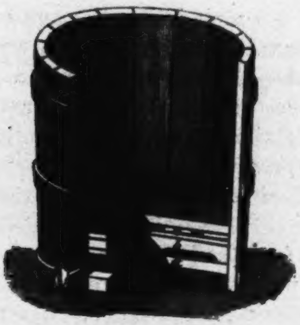
* At a temperature of 32° Fahr. the amount of chloride of silver dissolved by a solution of common salt is almost inappreciable. At 50° Fahr. a solution of salt takes up chloride of silver amounting to 0·0017 of the weight of chloride of sodium present, at 64° its dissolving capacity has increased to 0·0024, and about 212° to 0·0068, of the amount of salt in the brine.

Lixiviation and Precipitation.—The lixiviation of the matts thus prepared takes place in an upper story of the establishment, and is conducted in a series of tubs ranged in a line along one of its walls. The construction and form of these vessels will be understood by reference to Fig. 58, which represents one of the lixiviating tubs with the portion nearest the spectator removed. Upon the bottom *a* is first laid a wooden cross *b*, on which rests the perforated false bottom *c*, on the top of which is again arranged a layer of straw *d*; above this is placed a filter of linen cloth, made tight against the sides of the tub by means of a hoop, and on this is deposited the roasted matt to be lixiviated; the filtered liquor being drawn off by means of the tap *d'*. These tubs are three feet nine inches in height, two feet eight inches in diameter at top, and two feet four inches at bottom.

Fig. 58.



Lixiviating Tub.

Eight of them are charged at the same time, each with eight hundred pounds of the prepared matt; and a stream of hot brine is directed, through a perforated cover, into each, by means of suitable pipes arranged for that purpose. The chloride of silver being thus dissolved in the hot solution of chloride of sodium, is carried by it through the filter, and flowing off by the taps *d'*, is conducted through a wooden gutter to a covered tank, in which the particles of ground matt, carried off in suspension, are allowed to settle. The liquors from this settling tank are now conveyed to a series of three tubs, which, like those already described, are all furnished with false bottoms and filters; being also so placed one above the other that the liquor escaping from the tap of the first will flow into the second, and from thence into the third and last vat. In the bottom of the two upper tubs is placed a layer of cement copper from six to seven inches in thickness, whilst on the filter of the last is laid a quantity of scrap iron. On admitting the saline liquor into the first of these tubs, a large proportion of the silver which it contains is precipitated in the metallic form at the expense of the copper, which, by combining with the chlorine of the chloride of silver, is itself in turn converted into chloride. The liquors escaping from this tub are received into the second of the series, also provided with a thick layer of cement copper, by which the last traces of silver are precipitated. The liquid, which has been thus freed from silver, but has now become highly cupreous, falls directly into the lowest tub, containing scrap iron, by which the copper is deposited in a form suitable for the precipitation of silver in the upper tubs during succeeding operations.

The brine flowing from the last tub, and deprived both of its silver and copper, is now pumped into a proper receiver, and, after being boiled, may be again used for lixiviation. In this way the hot brine is passed through the lixiviating tubs until a plate of bright copper does not become coated with silver when held for some time in the liquors escaping from them. The residual matters remaining in the tubs, and which chiefly consist of oxide of copper, are then removed to give place to fresh charges, but before being treated for copper are assayed, in order to ascertain the amount of silver which they still retain. Should they be found to contain more than a quantity corresponding to 0·0003 of the weight of matt operated on, they are again roasted and lixiviated; but if, on the other hand, the proportion of silver be less than that stated, they are at once fused for copper.

The cement silver, in the form of a crystalline powder, is about once a week removed from the tubs in which it has been precipitated; and after being treated with dilute hydrochloric acid, for the purpose of removing any traces of copper, is thoroughly washed with clean water. When sufficiently washed, the metallic sponge is pressed into balls, thoroughly dried, and taken to the Saxon mint, where it is purified and fused into bars. This process is at present only employed for the extraction of silver from matts in which that metal has, in conjunction with copper, become concentrated by repeated roastings and fusions. It may, however, under certain conditions, be found directly applicable to some of the various ores of silver.

ZIERVOGEL'S PROCESS.—Shortly after the adoption of Augustin's process at the works belonging to the Mansfeld Company, the cheaper and still more simple method of converting the silver into sulphate, and subsequently dissolving it out by hot water, was introduced by Hüttenmeister Ziervogel, and since the year 1857 the whole of the argentiferous matts worked in the establishment have been treated according to this plan.

The efficiency of the method of Ziervogel depends on the circumstance, that when a finely-powdered matt, consisting of the sulphides of copper and iron containing a certain proportion of silver, is, with proper precautions, roasted in a reverberatory furnace, the iron and copper first pass into the state of sulphates, which are afterwards transformed into oxides. The sulphide of silver subsequently undergoes a similar transformation, and, if the roasting were continued, would ultimately be reduced to the metallic state. If, however, the operation be arrested at the proper stage, the copper and iron will have become transformed into oxides, whilst nearly the whole of the silver exists as a soluble sulphate readily removed by water; which thus affords a means of separating that metal from the other constituents of the charge, which are, for the most part, insoluble in that menstruum. From the argentiferous liquors thus obtained, the silver is afterwards precipitated by means similar to those employed in the method of Augustin; but when a solution of the sulphate is effected by the use of water, in place of dissolving the chloride by means of hot brine, nearly the whole of the silver originally present in the sulphides treated may be obtained in the metallic state.

Roasting.—The matt, after being ground between a pair of millstones, four feet in diameter, made of the granite of the Hartz, is bolted through a circular sieve, of from 1,400 to 1,500 apertures to the square inch, and then carefully roasted in a reverberatory furnace, specially adapted to the purpose.

This furnace, which is provided with two distinct hearths, placed one above the other, has an exterior length of nineteen feet, and a width of thirteen feet; its total height being about fifteen feet. The two hearths on which the roasting is conducted are each ten feet long by eight feet wide, and are built of good firebrick, of which all the parts of the apparatus subjected to a high temperature require to be constructed. The fireplace is provided with a grate, from which the flame passes across a firebridge over the contents of the lower hearth, and, ascending a perpendicular flue, the heated gases are conducted, in a tortuous direction, through a number of channels over the arch of the upper chamber. In these flues are deposited the fine particles of matt mechanically carried over by the draught, and which are, from time to time, removed through suitable openings, for the purpose of being subjected to metallurgical treatment. From these horizontal flues the gases finally pass off into the chimney, whilst the upper hearth, being heated both from above and below, is in the condition of a muffle, through which none of the gases of the furnace are allowed to pass. This hearth is, therefore, well adapted for effecting the calcination of ores when they cannot be brought to a high temperature without danger of fusing. In the bottom of this hearth, passing through the arch of the lower one, is an aperture, closed by an iron plate, through which the charge can be raked into the lower bed of the furnace. When the roasting on the lower hearth is completed, the charge is raked through another opening, also covered by a plate, into an iron waggon, which is run into an arched tunnel beneath it.

The ground matts from the mill are mixed with about 15 per cent. of the residues remaining in the tubs, after the process of lixiviation; and from 500 to 600 lbs. of the mixture is charged upon the upper hearth of the furnace, where it is exposed to a gradually increasing temperature, and kept constantly stirred during about seventy-five minutes. The portion of the charge most remote from the fire is now brought forward towards the bridge, and any lumps which may have been formed are carefully broken with an iron rod. The roasting is continued, as before, during another period of seventy-five minutes; and at the expiration of two and a half hours from the first introduction of the charge, an addition is made of from 20 to 25 lbs. of ground lignite, in a perfectly dry state. This is mixed with the pulverised matt, by a stirring which occupies about ten minutes, when the plate is removed from the hole in the bottom of the muffle, and the whole of the contents of the upper chamber are raked through it, and spread evenly over the surface of the lower hearth. The heat applied is, at first, moderate; but at the expiration of little more than an hour, during which time the charge has been constantly stirred, and the powdered lignite has become consumed, the temperature is considerably increased.

At the expiration of about ten hours from the time of first charging, the workman, who, particularly during the latter stages of the operation, has kept the contents of the hearth carefully stirred, withdraws a sample, for the purpose of determining whether or not the roasting has become sufficiently advanced. This is done by taking, with a small iron ladle, a spoonful of the ore from different parts of the hearth, and, after placing it in a white earthenware saucer, dropping on it a little pure water, which is allowed to percolate slowly through the edges of the sample. In this way the water takes up any soluble salts which may be present in the roasted matt, and, from the colour assumed by the solution, a correct judgment can, after a little practice, be formed of the condition of the charge.

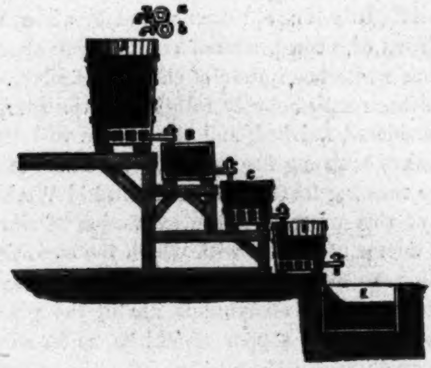
The liquor which thus escapes from the sample of roasted mineral should present a slightly blue appearance, but be without any shade of green: the addition of a few grains of common salt should, moreover, produce a copious white precipitate of chloride of silver. When these conditions have been satisfactorily fulfilled by the sample taken, the operation is considered finished, and the charge withdrawn. Brushwood, which makes a strong fire, with a long flame, is the only fuel employed in the roasting furnaces at the Mansfeld Works.

The success of this process for the extraction of silver manifestly depends on the degree of facility with which the operation of roasting may be controlled, so as to be enabled to seize the exact period at which the several metallic compounds are in the precise condition required. The sulphate of copper should be, as far as possible, converted into an oxide, whilst the whole of the silver ought to exist in the form of a soluble sulphate. Should the roasting be arrested before this point has been attained, a large amount of copper will be found to remain in a soluble state; whilst a portion of the silver still exists in the form of an insoluble sulphide. If, on the contrary, the roasting be carried too far, the sulphate of silver will have become reduced, leaving that metal in the metallic state; which, being totally insoluble in the hot water employed for lixiviation, will remain with the copper, and become commercially lost. Long practice and much observation are required on the part of the workmen employed in this process, and, with a view of increasing their efficiency, a scale of premiums has been instituted for any more than ordinary success in the work performed.

This system of rewards has been particularly successful at Mansfeld; for, although the process has not itself been in any way modified since its first introduction into the works, yet the skill of the workmen has, within the last few years, increased, in so marked a degree, that the results now obtained are much more satisfactory than those formerly yielded by this method of treatment.

Lixiviation and Precipitation.—The roasted argentiferous matt is now taken to the lixiviation department, which consists of a large room, in which a number of vessels are arranged (as shown in the following woodcut, Fig. 59), and so placed that the liquors flowing from one are immediately received in the next which follows in the series.

FIG. 59.



ZIERVOGEL'S PROCESS.
(*Arrangement of Apparatus.*)

The powder to be operated on is divided into parcels, weighing 400 lbs. each, which are placed in the vessels A, two feet six inches in diameter, and of about the same height, provided with filters and false bottoms; twenty-two cubic feet of liquor from a previous operation, together with about three cubic feet of fresh water, both heated to a temperature of 160° Fahr., are run into each of the upper tubs through the pipes *a*, *b*. A little sulphuric acid is also employed for the purpose of preventing the inconveniences which are found to accompany the presence of basic salts. This fluid, soon permeating the ore in the tubs A, takes up the sulphate of silver, and any other soluble salts present, which, passing through the filter, are carried in solution into the tank B, thirty feet in length, and eighteen inches square, divided into two parts. In this reservoir the liquors enter the first division, and, after allowing the matters held in suspension to settle, the solution flows over the partition, and from thence through ten taps into as many tubs C: in the bottom of each of these are placed 10 lbs. of cement copper and 250 lbs. of coarse copper bars, by which the larger proportion of the silver is precipitated in the metallic form. The fourth vessels D, of which there are five, also contain metallic copper, and in them are precipitated any traces of silver which may have escaped precipitation in the tubs C. From these last tubs the spent liquors flow off into the lead-lined cistern E; from which they are subsequently raised by steam pressure into another leaden cistern above the level of the first series of tubs A, heated to a temperature of 160° Fahr., and passed over a fresh charge of roasted matt, introduced into the series of dissolving vessels A.

About two and a half hours are required to dissolve out the sulphate of silver contained in each charge; and at the end of that time the residual contents of the dissolving tubs are transported to an adjoining room, where an assay sample is taken. Should the results of this assay show that the amount of silver remaining is less than 0·00036 of the weight of the material operated on, the residues are placed aside, for the purpose of being fused for blistered copper; but if, on the other hand, they contain more than this proportion of silver, they are re-roasted by the workmen, without any further payment for labour. When, on the contrary, the assay shows a less amount of silver in the residues than that above stated, the roasters receive a bonus of the value of 12 per cent. on the excess of the precious metal extracted.

The finely-sifted matt, after being withdrawn from the furnace, is allowed to remain about eight hours before being introduced into the lixiviating tubs, and thus becomes cooled down to about 160° Fahr. before charging. When placed in the tubs, hot water is admitted from *a*, until it begins to escape from the taps at the bottom. The water is then turned off, and hot liquors from a previous operation are introduced from the leaden cistern by the pipe *b*, until the liquid flowing from the cocks at the bottoms of the tubs no longer affords a precipitate of chloride of silver on the addition of a weak solution of common salt. The final liquors collected in the vessel E, when they have become too highly charged with sulphate of copper, are brought in contact with scrap iron, and thus afford a supply of cement copper, which may be subsequently employed in the tubs C and D.

The process of Ziervogel is, however, adapted to the requirements of comparatively few localities, since the presence of certain impurities, and particularly of any considerable amount of either arsenic or antimony, gives rise to the formation of insoluble salts, which materially interfere with the extraction of silver. In the Freiberg works, where the process was for some time experimented on, it was found that the presence of these substances so far interfered with the results obtained, that a notable amount of the silver present in the concentrated matts, invariably remained in the insoluble cupreous residues.

As might be anticipated, by far the largest proportion of the silver is deposited in the first precipitating vessel, from which it is from time to time taken for the purpose of being purified and melted into bars. The principal impurities with which it is associated are the sulphates of copper and lime; together with a certain amount of metallic copper derived from the cement copper employed as a precipitant. The two former are removed by repeated washings with hot water, whilst the latter is partially dissolved out by treating the finely divided silver with dilute hydrochloric acid. The precipitate is subsequently refined in a furnace constructed for that purpose, and affords bars containing about 980 thousandths of silver.

According to Lamborn, who carefully examined this process at the Mansfeld works, the cost of treating one hundred weight of copper, usually containing nearly ½ lb. of silver, at that establishment, was as follows :—

By Liquation	10	thalers.
„ Amalgamation	5½	„
„ Augustin's method	4½	„
„ Ziervogel's „	2½	„

The amounts of silver remaining in the Mansfeld copper after treatment by the several processes are the following :—

After Liquation	$\frac{1}{10}$th	of one per cent.
„ Amalgamation	$\frac{1}{17}$th	„
„ Augustin's method	$\frac{1}{17}$th	„
„ Ziervogel's „	$\frac{1}{84}$th	„

VON PATERA'S PROCESS.—This method of extracting silver from its ores consists—1st, in roasting them with an addition of common salt until the whole of the silver has been transformed into chloride; 2nd, in dissolving out the chloride of silver by means of a cold dilute solution of hyposulphite of soda; 3rd, in precipitating the silver in the form of sulphide by the addition of polysulphide of sodium; and 4th, in reducing the precipitated sulphide of silver to the metallic state by exposing it in a muffle, at a high temperature, to the ordinary influences of atmospheric air.

The solution in hyposulphite of soda of the silver contained in argentiferous ores was first suggested in a paper published by Dr. Percy in 1848; and a translation of this having reached the Austrian chemist, it resulted, in 1858, in the introduction, at Joachimsthal, by Von Patera, of the process which now bears his name. The ores from that district are remarkable for the diversity of their constituents, and in addition to silver frequently contain various compounds of copper, lead, bismuth, iron, nickel, and cobalt, associated with sulphur, arsenic, and antimony. The veins in the vicinity of Joachimsthal, though less productive than formerly, still afford a certain amount of argentiferous ore of extraordinary richness, since the average yield of the whole quantity delivered at the works may be taken as affording about 2 per cent. of silver. Smaller quantities are not unfrequently worked containing from 5 to 6 per cent. of that metal, and as much as 15 per cent. of silver has sometimes been extracted from a parcel of ore.

The fuel employed consists of lignite, coal, charcoal, and wood; the first of which is cheap and the second expensive, while the two last are becoming scarce, and growing every year higher in price. The price of labour varies from a shilling to one and sixpence per diem.

Roasting.—The ores, which are prepared partly by hand picking and partly by concentration on shaking tables, are, on being brought to the works, subjected to a process of roasting in a furnace of a somewhat peculiar construction. This apparatus, instead of having the long narrow hearth, broad firebridge, and short wide fireplace, usually employed for roasting sulphurous ores, has a hearth 9 feet 9 inches across, and measuring but 6 feet from the bridge to the flues leading into the chimney. The grate, which is only six inches in width, is four-fifths the length of the longer axis of the hearth, from which it is divided by a sort of firebridge, consisting of an iron tube covered with clay, and pierced with from ten to twelve small openings on the side furthest removed from the fuel. A small boiler, set in brickwork near the furnace, supplies low pressure steam, which can, when required, be introduced into the tubular bridge, and allowed to escape in numerous jets over the surface of the roasting ore.

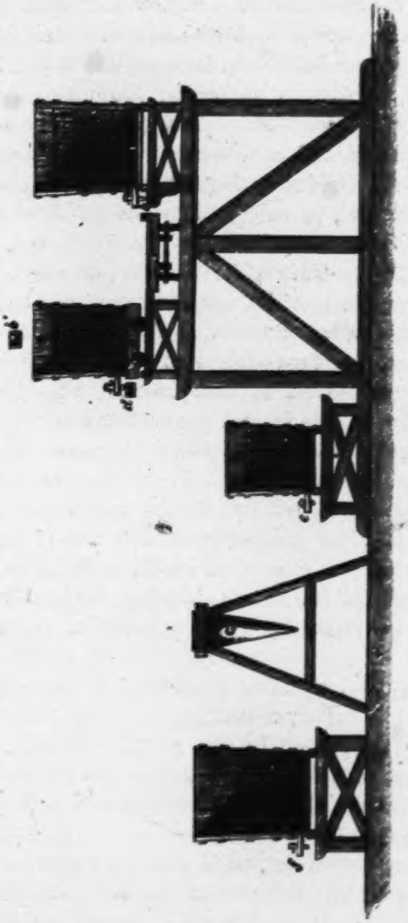
The mineral to be operated on is introduced into this furnace in charges of four hundred pounds, and the heat slowly and cautiously raised, in order to prevent agglomeration of the particles. During this stage of the operation no steam is admitted, but, as soon as the charge has arrived at a red heat, the tap is turned, and as much steam blown into the hearth as can be safely introduced without so far reducing the temperature as to materially check the activity of the various chemical decompositions which it is desired to effect. At the expiration of four hours from the time of charging, the operation is usually completed; and the ore, after being withdrawn and allowed to cool, is taken to a mill, in which it is ground to a fine powder, with the addition of from six to twelve per cent. of common salt, and two to three per cent. of sulphate of iron. A charge of this mixture, weighing three hundred pounds, is now introduced into a furnace similar to that above described, and is spread evenly over the surface of the hearth. This is raised to a red heat, and the steam admitted as before, care being taken to keep the contents of the apparatus constantly stirred. The temperature is now gradually increased, and at the end of from ten to sixteen hours, according to the nature and richness of the ore, the operation is complete.

The addition of sulphate of iron to the partially desulphurised ore is for the purpose of effecting the necessary decomposition of chloride of sodium, in case a sufficient amount of metallic sulphates should not be otherwise present. The introduction of aqueous vapours is found to facilitate the chlorination of the silver, besides greatly assisting in the condensation of the fumes and vapours in the chambers interposed between the furnaces and the chimney through which the products of combustion finally make their escape. If this recovery of the substance carried off by the draught were not carefully attended to, the loss of silver so caused would amount to about 10 per cent., and the economical treatment of argentiferous ores, by this process, be thus rendered impracticable. The roasted and finely-divided ore, containing silver in the state of chloride, is now taken to the lixiviating room for further treatment.

Lixiviation with Water.—The apparatus employed for the purpose of solution and precipitation will be understood by reference to Fig. 60, which represents a vertical section of the whole arrangement.

In addition to chloride of silver, which is insoluble in water, the ores contain a certain amount of copper, zinc, nickel, cobalt, iron, &c., which, being present in the form of sulphates and chlorides, are readily dissolved in that menstruum. Into each of the tubs A, in the first series, are introduced four hundred pounds of roasted ore; and

Fig. 60.



Von Patera's Process.
(*Arrangement of Apparatus.*)

boiling water is allowed to percolate through the several charges during a period of six hours. By this means all the soluble salts enter into solution, and passing through the filter *a*, are conveyed by the trough *b* into suitable tanks, in which they are precipitated by lime-water, and, if found to contain a sufficient amount of silver, are subsequently treated by fusion with lead ores in a blast furnace.

The liquors falling into *b* are from time to time tested by sulphide of ammonium, and, as soon as no further precipitate is obtained on adding to a sample a few drops of this reagent, the operation is considered to be finished, and cold water is passed through the tubs for the purpose of reducing the temperature of the residues; which must not, until quite cold, be subjected to the action of the solution of hyposulphite of soda.

Lixiviation with Hyposulphite of Soda.—The pulverised ore remaining in the several vessels A, which has been thus freed from all the different salts soluble in hot water, is now transferred to the tubs B, which, like the first, are furnished with filters and false bottoms. Seven of these are employed at Joachimsthal, placed on a level with the tub A, between which and the vessels B is a small railway on which is the car *c*. The tubs B stand on a low truck *c′*, which can be run from the position shown in the woodcut on to the waggon *c*, and afterwards made to traverse, either backwards or forwards, parallel with, and in close proximity to, the line of tubs A. The vessel B, after receiving a charge of two hundred pounds of the residual ore from one of the tubs A, is taken back to its place and there treated with the liquor by which solution of the silver is to be effected.

This consists of a cold aqueous solution of hyposulphite of soda, brought from a tank by means of the trough *b′*, and allowed to percolate slowly through the mass. In this way the chloride of silver is taken up in the form of a double salt and passes through the filter, in the bottom of the tub, into the trough *d*, by which it is conveyed to the precipitating tubs E, F.

The time necessary for the completion of the operation is more or less influenced both by the richness of the ores and their state of mechanical division; the richest samples, containing 15 per cent. of silver, requiring as much as forty-eight hours before becoming sufficiently impoverished; whilst the poorer ores, affording about 1 per cent. of silver, generally require but twelve hours for their treatment. In the case of ores not containing above 7 per cent. of silver, one chlorination and lixiviation is found sufficient, but when richer ores are operated on it becomes necessary to have recourse to two distinct processes of lixiviation, together with an intermediate roasting with salt and sulphate of iron. The lixiviation is known to be complete when the liquors dropping from the tubs no longer afford any traces of a precipitate on the addition of a few drops of sulphide of ammonium, and the residues are then removed, and, after being dried, fused in a blast furnace for copper.

Precipitation of Silver.—The liquor flowing through the filters in the tubs B, are conducted by the trough *d* into the vessels E, F, of which there are ten—six holding 40 gallons each, and four of the capacity of 80 gallons. The precipitant here employed is a polysulphide of sodium, produced by fusing common soda-ash with sulphur, and subsequently boiling the product, dissolved in water, with sulphur in a finely divided state. The solution thus obtained is taken in large stone jars to the precipitating tubs, and poured into the argentiferous liquors so long as a precipitate is produced by the introduction of an additional quantity. The contents of the tubs, after being well stirred, are allowed to settle, and a sample of the clear liquor having been taken in a test-tube, a little of the solution of sulphide of sodium is added.

If a dark-coloured precipitate be formed, it shows that a portion of the silver still remains in solution, and a further supply of the alkaline sulphide is required in the precipitating vessels. If, on the contrary, the addition of polysulphide of sodium has not the effect of producing a dark precipitate, it becomes probable that too large an amount of the sulphide may have been added to the argentiferous liquor. In order to ascertain this fact, some fresh liquor, holding the double salt of silver in solution, is added to a sample taken from the tub under examination. Should a precipitate of sulphide of silver appear, fresh argentiferous liquor must be carefully added to the tub until no further reaction is observed. When this point has been attained, all doubt as to whether the whole of the silver has been precipitated, on the one hand, and that no excess of the precipitant has been employed on the other, is removed by the addition, to one sample, of a few drops of a weak solution of common salt, and, to another, of a small quantity of acetate of lead.

If no precipitate of chloride of silver be produced by the addition of chloride of sodium, it is a proof of that metal having been completely removed; and should no discoloration take place on the addition, to the other sample, of a solution of acetate of lead, it shows that no excess of the precipitant has been added.

The exact neutrality of the residual liquid is essentially necessary to the success of this process, since the liquors from which the silver has been thrown down are employed in succeeding operations. It is evident therefore that the presence of sulphide of sodium would convert a portion of the silver into an insoluble sulphide of that metal, whilst any chloride of silver allowed to remain in the hyposulphite solutions would diminish their solvent properties, in a proportionate degree with regard to that salt. This process of precipitation, which would appear to be both delicate and complicated, is for each tub readily accomplished in the course of fifteen minutes by two workmen; and experience enables them to perform this operation with such facility that they never fail in obtaining the result desired.

Six hours are now allowed for the flocculent precipitate to settle at the bottom of the tubs, after which the clear liquor is syphoned off into a reservoir beneath the floor, and the black slimy sulphide drawn off by the taps *e*, *f*, to be placed in a filter-bag of close canvas.

It may be remarked that, instead of any loss of hyposulphite of soda taking place during the working of this process, an actual increase of that salt is the result. This is occasioned by the oxidation from exposure to the air, of the sulphide of sodium employed as a precipitant; and consequently the solutions containing this solvent have occasionally to be reduced in strength by the addition of water.

The spent liquors from which the sulphide of silver has been precipitated are afterwards pumped from the tank beneath the floor of the establishment to another above the level of the row of tubs A, from which they are drawn off, as they may be required, for the lixiviation of a subsequent charge of roasted ore.*

Treatment of the Sulphide of Silver.—The pasty sulphide of silver as drawn off from the precipitating tubs is placed in conical canvas bags G, supported on wooden frames, and allowed to drain. After standing in the filter until it has ceased to drip, the pasty mass, together with the enclosing bag, is placed under a screw press, and the remaining moisture expressed as completely as possible. The precipitate is now removed from the bag, dried, and, after being replaced in the filter, is washed with hot water for the purpose of removing the adhering soluble salts, of which sulphate of soda is the chief ingredient. The sulphide of silver, thus purified, is again dried, and afterwards heated to redness in a muffle, through which a current of air is allowed to circulate. In this way the sulphur is almost entirely burnt off, and at the expiration of about two hours the entire mass has assumed the metallic condition.

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* A pound of crystallised hyposulphite of soda in solution in twenty pounds of water, is capable of dissolving out, at 58° Fahr., an amount of chloride of silver corresponding to 0·358 lb. of metallic silver.

The present wholesale price of hyposulphite of soda is about 16*l*. per ton.

This metallic residue is now fused, in charges of about 300 lbs., in large plumbago crucibles, and any traces of sulphur, which it may still retain, removed by the addition of metallic iron, with which it forms a ferruginous sulphide readily skimmed from the surface of the metal. A small quantity of a mixture composed of wood ashes and bone-ash is now thrown on the surface of the metallic bath, and this, on being carefully scraped off, leaves the fused silver in a condition suitable for casting into ingots. The silver is thus obtained at a total cost of 9*s.* 9*d.* per lb. or about 6*s.* less than the cost by the old method of fusion with lead ores.* Bars produced by this process usually contain from 980 to 985 thousandths of silver.

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* Bodeman.

CHAPTER XX.

CONCENTRATION OF THE PRECIOUS METALS IN METALLIC LEAD—SMELTING.

PRELIMINARY OBSERVATIONS—SMELTING AURIFEROUS SILVER ORES FOR MATTS—EXTRACTION OF THE PRECIOUS METALS IN BATH OF FUSED LEAD—CONCENTRATION OF SILVER IN MATTS, AND FUSION WITH ROASTED LEAD ORES, WITHOUT ADDITION OF IRON—REDUCTION OF UNROASTED LEAD ORES BY METALLIC IRON—CASTILLIAN FURNACE—REDUCTION OF PARTIALLY-ROASTED LEAD ORES BY METALLIC IRON—RESULTS OBTAINED AT WILDBERG.

THE amount of the precious metals extracted from their ores by the process of smelting, even including the silver obtained from the treatment of argentiferous galena, is small as compared with the annual production by amalgamation. This mineral, however, although, without exception, containing a certain proportion of silver, cannot be for that reason regarded as a silver ore; since the value of the lead which it affords is generally far in excess of that of the silver; and, consequently, galena must be considered as an ore of lead, affording variable quantities of the former metal.

In the various processes which have been adopted for the extraction of the precious metals from their ores by smelting, advantage is taken of the affinity which lead possesses for them when in a fused state; and consequently it performs, when in that condition, a similar service to that rendered by mercury at lower temperatures. From the great facility with which the sulphides of silver are converted into an impalpable powder, which, by floating off on water, becomes commercially lost, the mechanical concentration of silver ores is a very wasteful operation; and hence the solubility of sulphides of silver in other fused metallic sulphides is generally made use of as a means of concentration. The gold present in the ore is also concentrated in these fused metallic sulphides.

This observation, as far as it relates to silver ores proper, is also to a great extent applicable to argentiferous galenas in which the sulphide of silver, at least evidently to a great extent, exists in the form of a mechanical mixture, and consequently it is not generally advantageous to carry their mechanical concentration to a very advanced stage. From this circumstance, the processes employed for the metallurgical treatment of galenas rich in silver, differ materially from those made use of for smelting ordinary lead ores, and consequently the peculiarities of treatment adopted in some cases, in which the amount of silver present almost entitles them to be regarded as ores of that metal, will be noticed at some length. The details of the various methods for the reduction of gold and silver ores are, to a greater or less degree, modified in accordance with the resources of the districts in which they are employed, and consequently the processes adopted in different localities are so exceedingly numerous and complicated, that even a brief reference to all of them would occupy far more space than is at at our disposal for that purpose. We shall, therefore, confine ourselves to a description of some of the more important methods of treating these ores, as employed in various large and well-conducted establishments, and of such as can, at the same time, be taken as types of the several systems which they represent; although, in a more or less modified form, they may have been adopted in other localities. The presence of gold in an ore of silver in no way modifies its treatment in the furnace, since the former metal invariably accompanies the latter, and is ultimately separated from it by the process of parting.

Smelting Auriferous Silver Ores for Matts, and Extraction of the Precious Metals by Bath of Melted Lead.—The following description refers to this process as employed, some years since, for smelting argentiferous ores in Lower Hungary, and is given as a type of the system it represents.* The ores from the mines of Schemnitz and Kremnitz contain both silver and gold; those smelted from the former locality being generally more auriferous than those obtained from the latter. At the mines these ores are partially in the form of *Scheide-erz*, or cobbed ore, and partly in the state of slime, or *Schlich*; the latter being again classified into silver ores, containing more than two loths of silver per centner, or 20 oz. per ton; and pyritous ores, generally yielding much less than that amount.†

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* A valuable paper "On the Smelting of Silver Ores," as then practised on the Continent of Europe, by the late Mr. John Henry Vivian, F.R.S., was published in "Records of Mining," Murray, Albemarle Street, 1829.

† Loths of 271 gr. and centners of 123·80 lbs. avoirdupois.

The ores are, at the works, subdivided into three classes, viz.:

a.—Those containing less than three loths per centner.

b.—Those yielding from three to five loths per centner.

c.—Those containing more than five loths per centner.

This classification enables the operations at the smelting works to be conveniently regulated, as the matts obtained from the fusion of the poorer varieties are, after roasting, smelted with those containing a larger amount of silver, and a matt is thus produced, which, after being in its turn subjected to several successive roastings, is again smelted with still richer ores. The various operations at the smelting works succeed each other in the following order:—

A. *First Fusion—Roharbeit*—The charge consists of poor ores of the average yield of one loth of silver per quintal, or 10 oz. per ton. Slags from third fusion, with limestone, as flux.

Products.—Crude matt, or *Rohlech*, containing from three to four loths of silver per quintal; which, after repeated roastings, is fused in the second operation. The slags produced are subjected to no further treatment.

B. *Second Fusion—Reich-roharbeit*—The charge is made up of ores of an average produce of two and a half loths of silver per centner. Roasted matt from first fusion; slags from third fusion; limestone.

Products.—Rich crude matt, containing from eight to ten loths of silver per centner, and which, after roasting, is passed into the third process. Slags of no value.

C. *Third Fusion—Frischarbeit*—The charge consists of rich ores; roasted matts from second and third fusion; limestone. *Metallic lead added in tapping pan.*

Products.—Argentiferous and auriferous lead, subjected to cupellation; rich matt (*Frischlech*), which is roasted and re-smelted; and slags, which are employed in fusions A and B.

D. *Cupellation* of lead obtained from tapping pan.

A. *Crude Smelting.*—The furnace in which the first fusion is effected is called a *Hohofen*. It is 18 feet 6 inches in height, and receives its blast through two tuyeres, supplied with air, either by bellows, moved by water-power, or from a blowing-machine, formed of an oblong box, provided with the necessary valves, in which a piston is made to traverse. The width of this furnace is 2 feet 9 inches, and its depth at the tuyeres 3 feet 3 inches; whilst the back wall slopes in such a way that at the top, or charging place, the internal depth is four feet five inches. Over the top of this furnace arched flues or chambers are constructed, for the purpose of collecting any fine particles of ore that may be driven over by the blast; which are occasionally removed through openings closed by iron doors. The tuyeres are placed exactly in the angles, and their direction is so adjusted that the blasts from them cross each other at a distance of about two feet from the back wall of the furnace. These tuyeres are made of wrought iron, and project about three inches into the furnace; the nozzles being covered with refractory clay to secure them from the action of the fuel and slags. The tuyeres have a very trifling inclination, and one of them is placed somewhat higher than the other; the lower tuyere being situated about two feet above the top of the breast stone. The well, formed in the bottom of the furnace for the reception of the fused matts, communicates with a breast pan, on each side of which is a tapping hole leading to two pans sunk in the floor of the establishment. The bottom of the furnace and lining of the breast pan are composed of brasque, which, after having been well rammed in by proper beaters, is subsequently cut out into the required form. The material operated on chiefly consists of pyrites, and stamped silver ores of very low produce, to which are occasionally added some low quality cobbed ores. These are charged in about the following proportions :—

Stamped pyrites ore about 70 parts; low class silver schlich (*a*) about 25 parts; poor cobbed ore 5 parts. This mixture is so composed that its average yield of silver may be something over one loth per centner, or 10 oz. per ton. The gangue, being chiefly silicious and argillaceous, is exceedingly difficult to fuse, and consequently requires a large proportion of flux, particularly in the first operation; in the subsequent processes the fusion is greatly assisted by the addition of roasted matts. The mixture consists of about 120 centners of ore, the same amount of slags from the third operation, and from 20 to 30 centners of limestone. The ores, fluxes, and charcoal are thrown into the furnace from an upper floor, and the matts collect in the well at the bottom. The slags flow off in a constant stream over the edge of the breast pan, and are thence taken away as being of no value.

As soon as the well and breast pan have become full of fused matts, they are tapped off into a float on the level of the floor, and usually contain from three to four loths of silver per centner, or from 30 to 40 oz. per ton. The furnace is kept in fire for about a month, when it is blown out for repairs, and during that time usually smelts about 130 tons of ore. Four men are employed at each furnace, and are paid in accordance with the amount of ore smelted. The crude matt is broken, and, after undergoing three successive roastings in heaps, is treated in operation B. For the purpose of being roasted, the matt is laid, in pieces of about the size of the fist, on a layer of wood; a little charcoal being mixed with the broken matt. Each roasting occupies from a week to ten days.

B. *Second Fusion—Reich-roharbeit.*—This operation is generally performed in the *Halbhohofen*, which is 11 feet in height, and at the tuyere, 2 feet in width by 3 in depth; at the top it is about two inches wider, and, from the inclination of the back wall, about nine inches deeper. This apparatus has only one tuyere. The high furnace is also employed for this purpose when new, and before its internal dimensions have become too much enlarged by the erosive action of the fused matts and slags.

The charge chiefly consists of poor washed silver ores (*a*) containing from two to three loths of silver per centner, of some cobbed ore of about the same produce, and a small quantity of pyritous slimes, so that the average yield may be about two loths per centner, or 25 oz. per ton. The relative proportions of these substances in the charge are nearly as follow:—

Washed silver ore about 70 parts, cobbed ore 20 parts, pyritous slimes 10 parts. To every hundred centners of this mixture are added from 40 to 50 of roasted crude matt, from 50 to 60 centners of slags from operation C, and 15 to 20 centners of limestone. The matt produced by the fusion of this mixture contains from 8 to 10 loths of silver per centner, or from 81 to 102 oz. per ton, whilst the slags are thrown aside as useless, unless there should not be a sufficient amount on hand, from process C, to serve as flux; in which case a portion of them may be reserved for that purpose. This furnace burns out in fifteen days, and in that time smelts about 40 tons of ore, together with a due proportion of slag and roasted matt. The fused sulphides obtained by this operation are roasted in precisely the same way as the matts from the first fusion, and then form one of the ingredients of the charge in process C.

C. *Third Fusion—Frischarbeit.*—This is conducted in the same furnace as the preceding, and the ore worked during a campaign, extending over a week, is about 16 tons. The mixture operated on consists of schlichs and the richest cobbed ores in the following proportions:—schlich about 40 parts, cobbed ore 60 parts. To every hundred centners of this mixture are added from 30 to 40 centners of roasted matt from the third fusion C, from 20 to 25 of that from the second B, and the same amount of limestone. From four to six centners of lead, in accordance with the richness of the ores, are also charged into the float in the course of each shift of eight hours. The matt from the furnace is tapped into the float on the top of the metallic lead, and well stirred, in order that the largest possible amount of the gold and silver contained in the fused sulphides may be made to combine with that metal. From its greater specific gravity the lead collects at the bottom of the pan; whilst the lighter matt floats on the surface, whence it is removed in the form of flat cakes in proportion as it cools. As soon as the whole of the matt has been taken off, and the lead begins to appear, the furnace is again tapped, in order that the metal may not become oxidised from exposure to the air, and in this way an additional quantity of the silver and gold is taken up by the lead at each successive tapping. The whole of the lead is not introduced into the float at the commencement of the operation, but at different periods during the progress of the shift of eight hours; at the expiration of which time the enriched lead is ladled out into moulds, and the next tapping is made into the float on the other side of the breast pan.

The amount of silver and gold present in the lead thus obtained obviously depends on the richness of the ores treated, but usually varies from 40 to 60 loths per centner—from 408 to 613 oz. per ton. The matts from this fusion contain about 14 loths of silver per quintal, or 143 oz. per ton; and, after being repeatedly roasted in small heaps, are added to the charge in a subsequent operation. The slags contain about one-fourth of a loth of silver per centner, 2½ oz. per ton, and are employed as a flux in operations A and B. The copper contained in the ore continues to accumulate in the matts until a regulus containing from thirty to forty per cent. is obtained, when it is removed for the purpose of being subjected to a special treatment for that metal.

D. *Cupellation.*—The cupellation of the rich lead, which is effected in a large circular cupelling furnace, with a marl bottom and a movable iron dome, will be described when treating of the extraction of silver from argentiferous lead.

The foregoing is a description of the various operations by which the precious metals were some years ago extracted from auriferous silver ores at Kremnitz, but we are not aware to what extent they may have since been modified.

CONCENTRATION OF SILVER, IN MATTS, AND FUSION WITH ROASTED LEAD ORES WITHOUT METALLIC IRON.—This process was extensively employed at Freiberg previous to the introduction of Augustin's method of extracting silver by means of a solution of salt. At present, the matts from certain descriptions of ore, after being so concentrated as to contain about 70 per cent. of copper and $\frac{42}{100}$ths per cent. of silver, are roasted with salt, and subsequently lixiviated with brine, as described page 409. This process of smelting was in principle very similar to that employed in Hungary. The separation of the metallic constituents of low produce ores was in this, as in the former case, effected by the production of matts; these, after repeated roastings, were smelted with richer ores containing lead, and the resulting argentiferous lead subjected to cupellation.

The blast furnace employed in Saxony for this purpose differed materially from the Hungarian, as its height, from the bottom of the brasque to the charging hole, did not exceed 9 feet. Its width at the top was 2 feet by 2 feet 9 inches in depth; but as it gradually expanded in descending, it was considerably larger in the vicinity of the tuyere and breast pan.

The ventilating channels, situated beneath the furnace and breast pan, were covered by slabs of gneiss, on which was placed a layer of broken slag, about six inches in thickness, overlaid by a bed of clay, which was well rammed, and had an inclination towards the front. When the furnace had been constructed on this foundation, the bottom and breast pan were formed by ramming in, with proper beaters, successive layers of a brasque composed of two parts of fireclay and one of powdered charcoal or coke dust, and afterwards cutting it out to the required form.

The breast was supported by a flat stone or cast iron plate, and was terminated, on one side, by an inclined plane of earth, over which the slags continuously flowed off; whilst on the other was an iron plate set on edge, in which was a slot for the tap hole, beneath which, on the level of the floor, was a float for the reception of the fused matt. The tuyere was of cast iron, placed in the middle of the back wall, and projecting from it about three inches; its degree of inclination depending on the nature of the ores to be operated on. The blast was supplied either by a blowing machine or a pair of leathern bellows set in motion by a water-wheel, and in the vicinity of each furnace was a trough of water, for the purpose of cooling the various tools employed.

Crude Smelting—Roharbeit.—The materials selected for this operation were silver ores of rather low produce, containing little or no lead, and less than five loths of silver per Saxon centner—about 45 oz. per ton. To these was added an equal weight of pyritous ores yielding only a quarter of a loth of silver per centner. This mixture was so proportioned that the resulting crude matt might contain about five loths of silver per centner; and the amount of matt produced was from 40 to 50 per cent. of the total weight of ore operated on. With the ores thus mixed, was smelted a little more than their united weight of slags from the fusion with lead. These slags, besides acting as a flux, usually contained from an eighth to a quarter of a loth of silver per centner.

In charging the furnace, the coke or charcoal had to be evenly distributed over the surface of the material which it already contained, whilst the ore was chiefly thrown near the back wall; care being at the same time taken to keep the upper part constantly clean, and a good nose of infusible slag around the end of the tuyere. When the ores contained blende, it was found desirable to volatilise a large proportion of the zinc, and with this view a high temperature was kept up during the whole progress of the operation. As however the use of a strong blast would entail the loss of a large proportion of any lead which might be present, ores containing that metal should, as much as possible, be excluded from all mixtures to be subjected to crude smelting. The various substances, in proportion as they became fused, collected in the bottom of the furnace and from thence passed into the breast pan, in the bottom of which the heavy argentiferous matts accumulated, whilst the lighter silicious slags floated on the surface, and, flowing over its edge, ran down the face of the inclined plane at the side.

When the pan had become sufficiently full of matt, a clay plug was removed from the tap hole, and it was drawn off into the float; in proportion as its surface became sufficiently cold, it was thence removed, in the form of discs about three inches in thickness. A smelter, one assistant, and a slag wheeler, relieved every twelve hours, were required to conduct the operations of each furnace.

The coarse matt, thus obtained, was broken into pieces of the size of an orange, and laid in heaps for roasting. These heaps were about four feet in height, six in width, and from twenty to twenty-five feet in length. They were placed under cover of a large shed; and as the matts contained a sufficient amount of sulphur to carry on combustion when the requisite temperature had been once attained, but little fuel was required for the operation. A layer of wood was first placed on the floor of the roasting shed, and over this was spread a little coal or charcoal, on which the broken matt was piled. Each heap, in the first roasting, continued on fire during about fourteen days, when it was opened, turned, and laid on a fresh bed of fuel, care being taken to place the most completely roasted pieces on the top and sides of the pile. Each heap was afterwards subjected to a third roasting, conducted in a similar way; except that, a large portion of the sulphur having been by this time expelled, a larger amount of fuel became requisite, and the time required for the operation was less considerable.

Smelting with Lead Ores—Bleiarbeit.—The ores subjected at Freiberg to this operation were classed as follows:—Silver ores free from lead, and containing above nine loths of silver per centner—about 90 oz. per ton; silver ores containing not less than 16 per cent. of lead; and lastly, all lead ores containing silver. The proportion of argentiferous galena was such that the whole mixture afforded from 30 to 35 per cent. of lead. These ores, after being well mixed, were roasted in a reverberatory furnace, until all smell of sulphur had ceased to be evolved, after which they were agglomerated in order to fit them for working in the blast furnace.

The furnace employed for this purpose precisely resembled that used for crude smelting, except that the dimensions of the hearth were somewhat larger. The tuyere also required to be kept more inclined than during the first fusion; and more particularly so, when the ores contained blende, since in that case the portions of the furnace in the vicinity of the hearth required to be kept at a high temperature. In order to prevent loss of lead it was also necessary to regulate the blast and the supply of fuel, in such a way as to keep the top of the furnace always dead, and the nose of the tuyere at a dull red heat.

To the mixture of calcined ores from the reverberatory furnace were added from 60 to 70 per cent. of roasted matt from the operation of crude smelting, and a small quantity of cupel bottoms and other furnace products containing lead. The average produce of the rich lead thus obtained was about two marks per quintal, or about 300 oz. per ton. The plumbiferous matt, *Bleistein*, obtained from this operation, which floated on the surface of the metallic bath in the tapping pan, was taken off in the form of thin cakes, and subjected to repeated roastings in heaps, after which it was either smelted by itself, or with the addition of poor copper ores. As, however, the ores invariably contained a small amount of copper, the products were in either case argentiferous lead, copper matts, and silicious slags; the slags both from this operation and from the fusion with lead ores being employed as a flux during the crude smelting, whilst the copper matt or *Kupferstein* was re-roasted and treated for that metal.

The furnace employed for this fusion was kept in blast about six days, and during that time smelted about seven tons of calcined ore, and from three to four tons of roasted matt. The argentiferous lead, *Werk-blei*, was formerly treated by cupellation.

Various modifications have of late years been introduced into the processes employed at Freiberg for the treatment of silver ores, and more particularly so since the suppression of barrel amalgamation, and the introduction of Augustin's process. A writer in the *Revue Universelle des Mines* states that, in 1863, nearly twenty-five thousand tons of ore were treated by fusion at the various works in the vicinity of Freiberg; about half these were lead ores, of which the poorest contained from 15 to 30 per cent. of lead, and the best from 30 to 80 per cent. The whole of the lead was present in the form of galena, which was associated with ordinary iron pyrites, arsenical pyrites, blende, quartz, and carbonate of lime. The ordinary tenure of these ores was about 40 per cent. for lead, and 48 oz. of silver per ton. In addition to lead ores, various argentiferous minerals, containing about 32 oz. of silver per ton, are smelted with plumbiferous ores at Freiberg. It is, however, considered necessary that these should not contain above 6 per cent. of copper, since ores containing more than 6 per cent. of that metal are treated apart; and very poor silver ores are enriched by fusion in reverberatory furnaces with a mixture of slags from the blast furnace. The matts thus obtained are called *Rohstein*.

Previous to smelting, the ore is roasted in a double-soled furnace, of which the upper compartment is heated in the same way as an ordinary muffle.

The mixtures are composed as follow:—

Roasted Ore	60 cwt.
Rohstein	20 „
Slags	100 „
Limestone or fluor spar	2½ „

In order to smelt this quantity of material, from 24 cwt. to 27 cwt. of coke are required, or from 40 to 45 per cent. of the ore operated on. The blast furnaces employed have each two tuyeres; and, at a height of 4 feet 6 inches above them, are divided, from front to back, into two parts by a brick division, nine inches wide, which is continued up to the charging hole. The height of the body of the furnace is about 12 feet 6 inches; the depth of the canal, which communicates with the breast pan, is 15 inches below the nozzles; the width of the back of the furnace is, at the tuyeres, 5 feet 6 inches, width of the breast at same level, 3 feet 5 inches. The depth of the furnace at this level is 3 feet 7 inches, and the distance between the tuyeres 23 inches.

Smelting is conducted with a dull charging hole, and a nose of from four to five inches in length is kept on each tuyere. In the ordinary way of working the furnace, it is charged with one measure (20 lbs.) of coke, and alternately, with two, and four, measures of the usual mixture, each measure of which weighs about 60 lbs. If the nose should become too short, four measures of mixture are added to one of coke; but if, on the contrary, it should become too long, the proportion of coke is augmented.

Three workmen are employed at each furnace, viz., a smelter, an assistant, and a man who wheels away the slag, &c. &c. Each furnace is tapped four times in the course of twenty-four hours, and at each tapping 4 cwt. of metallic lead are obtained, containing variable quantities of silver, and 2 cwt. of argentiferous matts, yielding 20 per cent. of lead, besides 10 per cent. of copper. In addition to this, 73 per cent. of the entire weight of the mixture charged is obtained in the form of slags, yielding from 5 to 6 per cent. of lead, and a small quantity of silver. These slags usually contain less than 30 per cent. of silica, and are, consequently, highly basic. Each campaign lasts about ten weeks. The lead is desilverised by crystallisation, whilst the matts, after being roasted, are re-smelted with an admixture of blast furnace slag and fluor spar, thus affording a copper matt containing 30 per cent. of that metal. All the slags from the fusion of ores, as well as those obtained from the treatment of matts, are subsequently treated for the purpose of extracting from them the last traces of metal; and at the same time effecting the fusion of ores, poor in silver, which contain neither lead nor copper. These are generally pyritous and arsenical, and a portion of them is first roasted until it contains only about 5 per cent. of sulphur, and is then charged into a reverberatory furnace with a slag bottom; the charge is composed as follows:—

Roasted Ore	5 cwt.
Raw „	5 „
Lead Slags	20 „
Total	30 cwt.

When necessary, from 100 to 200 lbs. of quartzose sand are added to this mixture. At the expiration of 2½ hours the whole charge is in a state of fusion, and the furnace, for a few minutes, is fired very sharply. In this way a matt is obtained which accumulates at the bottom, where it is allowed to remain, whilst the slags are removed, either by tapping, or by being drawn off by an iron rake. As soon as this has been done, a new charge is added, and the operation is repeated; the matt being tapped off only after the fusion of the third charge. The matt thus obtained from the three charges usually weighs from 1 to 1½ ton, and contains all the useful metals; whilst the slags, on the other hand, are exceedingly poor: this matt contains from 4 to 5 per cent. of copper, from 8 to 10 per cent. of lead, and from 45 to 60 oz. of silver per ton. This matt is subsequently broken into coarse fragments, roasted in heaps, and afterwards smelted in the blast furnace with roasted ores.

It may be remarked that, although the processes of smelting employed at Freiberg are doubtless well adapted to the peculiar conditions existing in that locality, they could only be adopted, with any degree of advantage, in districts affording cheap labour, with fuel at a reasonable price.

REDUCTION OF UNROASTED LEAD ORES BY METALLIC IRON.—The process of treating lead ores by metallic iron is chiefly employed for smelting galenas containing a considerable amount of silicious gangue. The reduction of galenas rich in lead, of which the matrix is chiefly calcareous, and which generally contain but a small amount of silver, is often effected by first roasting on the hearth of a reverberatory furnace, and a subsequent fusion in the same apparatus.

In this case the metal is obtained by the double decomposition which takes place between the undecomposed sulphide of lead and those portions of the ore which have, by roasting, been converted into oxide and sulphate of lead. In this way the decomposition of one equivalent of sulphide of lead, and two of the oxide of the same metal, give rise to the production of an equivalent of sulphurous acid, and the liberation of three equivalents of metallic lead—

$$PbS + 2PbO = SO^2 + 3Pb.$$

In the same way, the fusion together of one atom of sulphide of lead and one of sulphate, results in the formation of two equivalents of sulphurous acid, and the liberation of two of metallic lead—

$$PbS + PbO,SO^3 = 2SO^2 + 2Pb.$$

The presence, however, of a very small proportion of silicious or argillaceous gangue renders the application of this principle extremely difficult, since when contained in the ores to the extent of only from 12 to 15 per cent., no metallic lead can be obtained in the reverberatory furnace by these reactions. As before stated, lead ores containing a large amount of silver cannot be advantageously enriched by mechanical treatment beyond a certain moderate percentage; and it is consequently usual, in smelting such minerals, to fuse them either in the raw state, or, after a partial roasting, with an admixture of metallic iron. The iron in this case, by combining with the sulphur of the galena, liberates the lead, and sulphide of iron is produced. When ores only partially roasted are operated on in this way, the intermediate reactions are of a somewhat more complicated character; but the final results, as far as regards the liberation of lead, are the same.

At Clausthal, in the Hartz, the ores usually contain about 30 oz. of silver per ton, and the furnace mixture consists of hand-selected and washed ores, to which are added certain secondary products, and a small proportion of granulated cast iron.

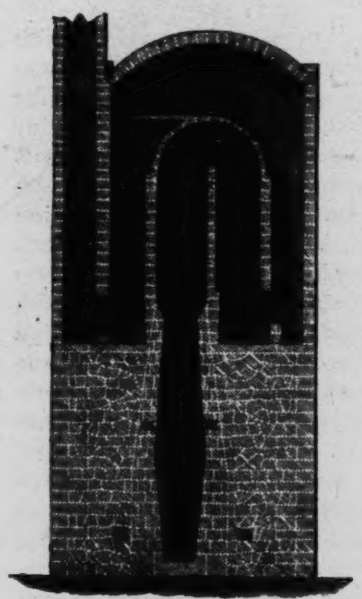
The charges are composed as follows:—

Hand-picked and washed ores, containing 55 per cent. of lead, 34 parts; old cupel bottoms, saturated with litharge, 4 to 5 parts; impure litharge from cupel, 1 part; slags from a previous fusion, or from the treatment of roasted matt, 39 parts; granulated cast iron 4½ parts.

The fusion is conducted in a blast furnace, varying from 18 to 20 feet in height, and having a width at the boshes of about 3 feet 6 inches, Figs. 61, 62, and 63: Fig. 61 being a vertical section at right angles to the tuyeres; Fig. 62 a section through one of the nozzles; and Fig. 63 a horizontal section above the breast pan.

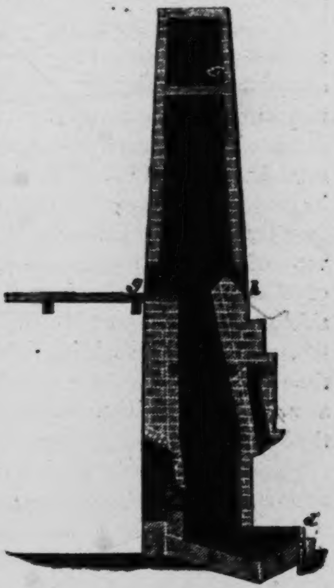
The casing walls are of common masonry, but the interior lining *a* is of firebrick; the bottom *b*, and breast stone *c*, are of sandstone, the breast pan itself being supported by iron plates *d*. The brasque of which it is formed extends under the front wall of the furnace into the interior, and a communication is thus established by which the metal and liquid slags can flow into the pan *e*; the tuyeres *f* are placed in the back wall, and have a slight inclination downward, and

Fig. 61.



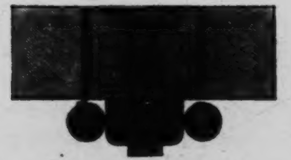
ClaustHAL Furnace.
(*Vertical Section.*)

Fig. 62.



ClaustHAL Furnace.
(*Vertical Section through Tuyere.*)

Fig. 63.



ClaustHAL Furnace.
(*Horizontal Section, through Tuyeres.*)

also towards each other. At top, on a level with the charging hole, the form of this furnace is circular; at the level of the tuyeres it is a parallelogram, with truncated angles; and at the bottom, a square. The distance between the back and front walls is, at the tuyeres, four feet, and the width, two feet six inches. The diameter of the charging hole is two feet.

This furnace is fed through the back by the opening *g*, and the small aperture *h* enables the workmen to instantly perceive the appearance of any flame which may escape at the top; it being essential, in order to avoid an undue waste of lead, to keep the charging hole constantly dead. The arrangement of the flues and condensing chambers is seen, Figs. 61 and 62. The side flue *i* is connected with the projecting arch or mantle, and is for the purpose of carrying off any arsenical or other vapours that may arise from the breast pan *e*. In the back wall of the furnace are small iron doors *k*, for the convenience of cleaning out the various deposit chambers and flues. On each side of the breast, on a level with the floor of the establishment, is a float or tapping pan *l*, into which the liquid metal is from time to time run out; whilst the slags flow off in a continuous stream from a notch in the iron plate, supporting the brasque which forms the breast.

The ores are chiefly charged along the wall in which are placed the tuyeres, whilst the fuel is principally spread in the vicinity of that which is opposite to them. The blast, passing through the tuyeres, and coming into immediate contact with the liquid slags in the body of the furnace, so cools them as to form a tubular elongation or nose. This is so managed by the smelter as to extend from five to six inches beyond the tuyere. When the basin and hearth have become full of matt and metallic lead, a plug of clay is removed, and their contents are run off into one of the floats *l*, where they are allowed to remain until the matt begins to solidify; when it is removed in discs, as before described, and the metallic lead ladled into moulds ready for desilverisation.

The matts thus obtained consist of sulphide of lead, associated with any other metallic sulphides which may have been originally present in the ore; together with sulphide of iron, resulting from the decomposition of galena by that metal.

The matts so produced are laid aside for subsequent treatment whilst the slags, except such as accidentally contain lead, either in the form of shot or in combination with silica, are thrown away as useless. Under ordinary circumstances a sufficient amount of rich slag is produced to serve the purpose of flux in the furnace mixtures; but when such slags are not obtained in sufficient quantities, poor ones have to be employed in their stead.

The matts are thus allowed to accumulate in the works, until a sufficient amount to enable their special treatment to be proceeded with has been got together; when, after being broken into pieces of a convenient size, they are roasted in heaps, each containing from 50 to 100 tons.

The first roasting occupies from three to four weeks, at the expiration of which period, the fire having burnt out, the pile is turned over, and the fragments which are found to be sufficiently roasted are picked out for the purpose of being smelted. The remainder, after this selection, is subjected to a second roasting, at the termination of which the heap is again picked over, and the residue subjected to a third roasting. In this way the insufficiently-roasted fragments remaining from one heap are made to form one of the constituents of another roasting, until the whole has become sufficiently desulphurised to allow of its being advantageously fused in the blast furnace.

The roasted matts are smelted in a small furnace called a *Krumofen*, which is much employed on the Continent for the treatment of substances requiring a moderately high temperature for their fusion. This furnace is supplied with air through a single tuyere, around which a nose, of from three to four inches in length, is allowed to grow. The bottom and breast are formed of brasque, the tapping being accomplished by the removal of a plug of clay from a channel communicating with the forehearth, which allows the fused metal to flow into a float sunk below the level of the floor. The charging takes place at the opening at top, and the fuel made use of, both in this furnace and in that employed for the first fusion, is coke; although charcoal or a mixture of coke and charcoal was formerly employed.

The mixture treated in the krumofen is constituted as follows:—Roasted matt, 32 parts; rich slags, 30 parts; cupel bottoms, 4 to 5 parts; impure litharge, 2 parts; reducing cinder, 2 parts; granulated iron, 1 part.

From this second fusion, as from the first, the results are matt and argentiferous lead. The matts from the second fusion are treated as before, and the same series of operations is continued until the fourth fusion. The sulphides which then result are known as copper matts, and, after numerous successive roastings, are smelted for coarse copper, from which the silver is generally extracted by liquation.

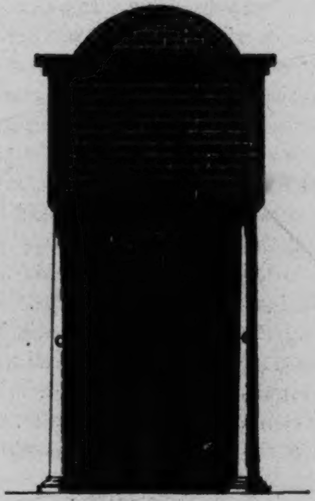
CASTILLIAN FURNACE.—REDUCTION OF PARTIALLY-ROASTED LEAD ORES BY METALLIC IRON.—This furnace, which was first employed for the treatment of Roman slags in the neighbourhood of Carthagena in Spain, is now used, not only for the reduction of various furnace products and argentiferous galenas of low produce for lead, but also for smelting, with the addition of plumbiferous matters, the rich silver ores imported from abroad.

This apparatus has a circular form, is generally about 2 feet 8 inches in diameter, inside measure, and is constructed of arched firebricks, so moulded as to leave no unnecessary spaces between the joints. Its usual height, from the floor of the works to the charging hole, is about 8 feet, and the thickness of the masonry the depth of one brick, or 9 inches. The bottom and breast are formed of brasque, supported in front by a curved plate of cast iron, furnished with a lip for running off the slags, and provided with a perpendicular slot, in which is situated the tapping hole. The details of this apparatus will be understood by a reference to Figs. 64, 65, 66, which respectively represent an elevation, a vertical section, and section at the top of the breast pan.

Around the top of the cylinder of brickwork A, is a hood of masonry B, supported on a cast iron framing, resting on four iron pillars C, in which is the door D, for feeding the furnace, and the outlet E, by which the various products of combustion escape into the flues. The lower part of this hood is fitted closely to the body of the furnace, whilst its top is closed by an arch of bricks turned on edge and set in fireclay. The hood B is built of ordinary non-refractory bricks set in lime. The bottom F is made of a brasque composed of one part of fireclay and two parts of coke dust, slightly moistened and well beaten in to the level of the breast pan G, which stands nearly three feet above the floor of the smelting house.

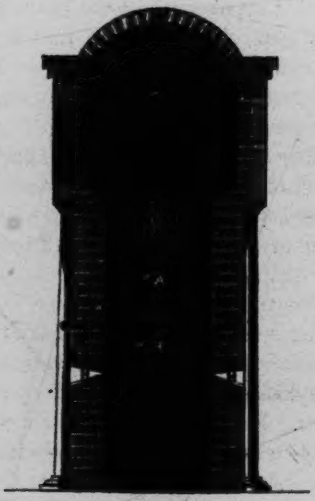
When the bottom has been solidly beaten in up to the required height, it is hollowed out so as to form an internal cavity communicating freely with the breast pan, which is itself hollowed out to a depth slightly below the level of the internal basin. The blast is supplied by three water tuyeres H, 2½ inches in diameter at the smaller end, 5½ inches at the larger, and 10 inches in length. Into these are introduced the nozzles of sheet iron, by which the blast is supplied by means of a fan, or ventilator, making about 800 revolutions per minute. The blast is generally conveyed to the nozzles through brick channels formed beneath the floor of the building.

FIG. 64.



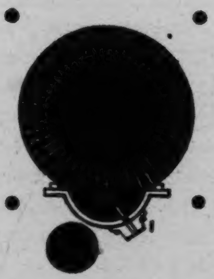
CASTILLIAN FURNACE.
(*Elevation.*)

FIG. 65.



CASTILLIAN FURNACE.
(*Vertical Section.*)

FIG. 66.



CASTILLIAN FURNACE.
(*Horizontal Section.*)

This furnace has not only the advantage of smelting a large amount of ore in a given time with a small expenditure of fuel, but also, from the thinness of its walls, the masonry does not become so intensely heated as to cause its rapid destruction. When any portion of this apparatus becomes burnt through, the damage may be generally repaired, either by plastering the outside with clay, or, if necessary, introducing a few new bricks. The Castillian furnace can therefore frequently be run from six to eight weeks before being finally blown out for repairs.

The ores treated in this apparatus may contain from 30 to 50 per cent. of lead, and require to be roasted either in heaps, in kilns, or in reverberatory furnaces, and are then generally smelted with an admixture of slags and limestone. In charging, the ores and fuel are supplied in distinct strata, and care is taken to so dispose the coke that the brickwork may not be exposed to the corrosive action of the slags. In order to allow of the free escape of slags the breast is not continued much below the level of the hearth, which, for the purpose of preventing the cooling of the scoriæ, is kept covered either by cinders or coke dust. From the breast, the slags flow constantly off by the lip I, into cast iron waggons, where they consolidate into masses having generally the form of truncated pyramids, of which the larger base is about two feet square. As soon as a sufficient amount of metal has accumulated in the bottom of the furnace, it is let off into a lateral float K, by removing the clay stopper of the tap hole, situated in the slot L of the breast pan; and, after allowing the matt to cool, it is taken off in flat cakes, and the still liquid lead cast into moulds.

The matt consists of sulphide of lead, more or less mixed with sulphide of iron, produced by the decomposition of imperfectly roasted ore by that metal; and, if the ores treated originally contained copper, a certain proportion of sulphide of copper will be also present. These matts are subsequently roasted in heaps, and afterwards smelted in the same furnace with an admixture of slags and limestone. The products of this fusion are argentiferous lead and matts, which after being subjected to repeated roastings, and being again passed through the furnace, contain so large an amount of copper as to be regarded as copper matts; and are sold, under the name of "regulus," to be treated for that metal.

The slags in the waggons are allowed to solidify, and, when sufficiently set, the casings are turned over, so that the blocks readily drop out. In addition to the facility for transport thus obtained, one of the great advantages derived from this method of manipulation arises from the circumstance that, should the furnace at any time run off either lead or matt without its being detected by the smelter, the greater portion of it will be found collected at the bottom of the blocks of slag, from which, when cold, it can readily be detached.

As before stated, in working these, like all other blast furnaces employed for smelting lead ores, great care must be taken to prevent the appearance of flame at the charging hole; since, provided the slags be sufficiently liquid, the lower the temperature at which the fusion can be effected the less will be the loss of metal experienced through volatilisation. In addition to great attention being paid to the working of the furnaces, it is essential, in order to obtain the best possible results, that the establishment should be provided with long and capacious flues, in which the condensation of the fumes may take place before arriving at the chimney. These flues should be built at least three feet in width and six in height, so as to admit of being readily cleaned out. They are sometimes made of very great length, since in one of the large establishments in the north of England their flues are now extended to a length of above seven miles, and a considerable amount of deposit takes place, even in the latter portions of its circuit.

In case of working finely pulverised ores in blast furnaces, it is essential that, after roasting, they should, before being withdrawn from the furnace, be agglomerated into masses, which are subsequently broken into fragments of the size of the fist, and then mixed with the fluxes before described. If the charge of a blast furnace contains much fine ore, or there be much coke dust in the fuel, the draught is affected, and the working of the apparatus becomes seriously deranged.

Each of these furnaces requires two smelters and an ore tender, who are relieved every twelve hours, during which time about three and a half tons of roasted ore, independent of slags and other fluxes, are passed through it, with a consumption of from 18 to 20 per cent. of coke. At the Wildberg Smelting Works in Prussia, where the ores raised consist of a fine-grained argentiferous galena, disseminated in a silicious gangue very difficult to smelt, and containing a considerable amount of spathose iron, the charge was, in 1859, composed as follows:—Roasted ore, containing by assay 42·8 per cent. of lead, 100 parts; slag from previous operations, 42 parts; limestone, 7 parts;

granulated cast iron, 8 parts.[*] The results obtained were about 39 per cent. of work lead, slags to about double the weight of the lead obtained, and 5 per cent. of lead matts, containing a certain amount of silver and copper. The roasted ores, at Wildberg, require 20 per cent. of coke for their treatment in the blast furnace.

Cobbed and other lead ores which have not been subjected to mechanical concentration may, before being smelted, either alone or with minerals containing silver, be conveniently roasted in kilns consisting of rectangular chambers 10 feet by 8 feet, having arched roofs, and provided with proper flues for the escape of the evolved gases, as well as with a wide iron door for charging and withdrawing the ores operated on. Each of these chambers will contain from 25 to 30 tons of mineral, and in order to charge it, a layer of faggots and cord wood is laid upon the floor, and this, after being covered by a layer of raw ore two feet in thickness, is ignited; care being at the same time taken to close, by means of loose bricks, the opening of the doorway to the same height. When the first layer has thus become well ignited, more ore, mixed with a little charcoal or a few small billets of wood, is thrown upon it; and when this layer has, in its turn, become sufficiently heated, another stratum of ore is added. In this way more ore is from time to time thrown on the top of that already in the kiln, until it has been filled to the spring of the covering arch, when the opening is closed by the iron door, and the operation proceeds regularly and without further trouble, until a large portion of the sulphur has been burnt off. This generally happens at the expiration of about four weeks from the time of first charging: the kiln then cools, the loose brickwork is removed, the roasted ores broken out with picks, and, after being mixed with scrap iron and the other necessary fluxes, they are passed through the blast furnace.

The following were the results obtained at the Wildberg Smelting Works during three months in 1859:—

Roasting:—

	Ctr.		£.	s.	d.
Roasted in kilns	10,679	cost per ton .	0	6	9
" in reverberatory furnace	3,321	" .	0	14	6

700 tons at an average cost of 9*s.* per ton.

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[*] The prices of materials delivered at the works were then as follow:—

	£.	s.	d.	
Granulated cast iron	3	18	0	per ton.
Coke	2	3	4	"
Limestone	0	15	0	"

Smelting:—

	ctr.		Lead. ctr.	Silver. oz.
Smelted	11,240 ores	containing	4,835	6,125
„	308 „	„	147	161
„	453 fahlerz	„	387	1,235
„	534 calcined dross	„		
	12,535 ctr.	„	5,369	7,521
		which produced	4,886	7,405
		showing a loss of	483* = 3⅞ units.	116†

Total Cost, at Wildberg, of treating 1 *ton of Ore, containing* 42·8 *per cent. of lead and about* 11 *oz. silver.*

	£.	s.	d.
Roasting	0	9	0
Smelting	1	7	1
Calcining	0	2	9
Crystallising	0	8	0
Reducing	0	2	3
Refining	0	0	9
	£2	9	10

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* In addition to the lead lost in smelting, about two units were volatilised in the process of roasting. The Wildberg ores, however, contain copper, antimony, and other impurities, and, consequently, the losses experienced in calcining and desilverising the hard lead, and converting it into soft marketable metal, are very considerable. The total loss on the assay produce of lead may consequently be taken at about 11 units.

† The difference of silver subsequently obtained from matts; the total annual yield being slightly above the amount indicated by assay, although a certain quantity always goes off in the market lead and in the matts sold as copper regulus.

CHAPTER XXI.

EXTRACTION OF THE PRECIOUS METALS FROM ARGENTIFEROUS AND AURIFEROUS LEAD OBTAINED BY SMELTING.

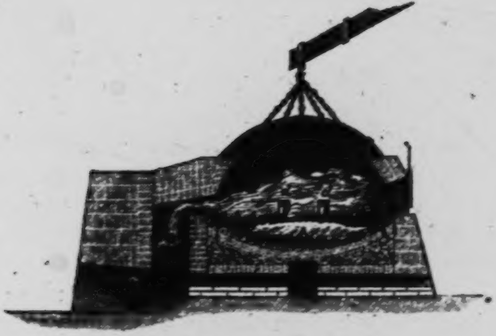
CONTINENTAL PROCESS OF CUPELLATION — REFINING SILVER OBTAINED FROM CUPEL—ENGLISH SYSTEM OF TREATING ARGENTIFEROUS LEAD—CALCINATION —CONCENTRATION OF SILVER BY CRYSTALLISATION—PATTINSON'S PROCESS—FRENCH PROCESS—REFINING ENRICHED LEAD—COSTS AND RESULTS OF DESILVERISING—LIQUATION.

Cupellation.—The argentiferous lead obtained by the treatment of ores of that metal, or the fusion of silver ores with roasted galena, or some other plumbiferous material, is either directly subjected to the process of cupellation, or the silver is previously concentrated in a small portion of the lead, by a series of crystallisations in a set of large cast iron pots or kettles.

The process of cupellation is, as before stated, dependent on the property possessed by lead of becoming oxidised when strongly heated, and at the same time exposed to the air; whilst the silver with which it is united is not so affected, and consequently becomes gradually concentrated in that portion of the lead which still remains in the metallic state. It results, from this difference in the behaviour of the two metals, that, when the operation has been continued for a sufficient length of time, the lead becomes entirely oxidised, whilst the silver remains in the metallic form. In order that the oxidation of the lead should proceed rapidly, and without interruption, it is necessary that the temperature to which the alloy is subjected should be sufficiently elevated to cause the fusion of the litharge produced. The surface of the fused alloy naturally assumes a convex form, and the fused oxide of lead constantly flows into the annular space existing between the edges of the metallic bath and the surface of the vessel in which it is contained. If, therefore, a depression be made in the edge of the containing vessel, so as to form a channel, the level of which can from time to time be lowered by scraping the bottom by means of a pointed iron bar, the whole of the litharge may be thus drawn off without the escape of any of the unoxidised argentiferous lead.

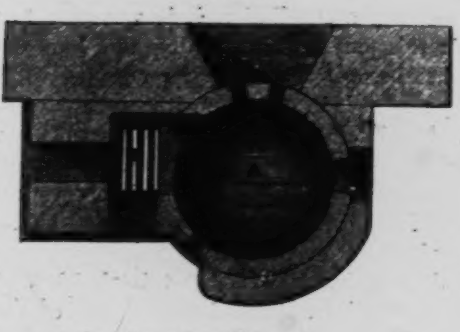
The furnace commonly employed for this purpose on the Continent of Europe, where the lead to be operated on is often subjected to cupellation without any preliminary concentration of the silver, is

Fig. 67.



Cupelling Furnace.
(*Vertical Section.*)

Fig. 68.



Cupelling Furnace.
(*Horizontal Section.*)

represented, Figs. 67 and 68, of which the first is a vertical section and the second a section at the level of the tuyeres.

This apparatus is a reverberatory furnace, consisting of a circular hearth from 9 to 10 feet in diameter A, sloping from all sides towards the centre, built of bricks *b*, set on edge on a layer of broken slags *c*.

On this is laid a bed of marl, *a*, which is firmly beaten in, when in a damp state, and renewed after each operation.* This bed of marl, in point of fact, constitutes the cupel, which is heated by means of faggots of brushwood burnt in the fireplace B. The roof of this furnace consists of a sheet iron dome C, which may be suspended by chains from the crane D, by which it can be removed and again placed in its position over the cupel. This dome is internally plastered with clay.

The cupelling furnace has five openings: one through which the flame from the grate enters the hearth; two, *d*, through which the nozzles pass supplying the blasts to the surface of the metallic bath, for the purpose of oxidising the lead, and which also assist in carrying the resulting litharge towards the annular space before referred to: the aperture E is employed for the introduction of the discs of lead to be cupelled; and F is that through which the fused litharge makes its escape from the furnace. At the commencement of the operation this last opening is closed by the edge of the layer of marl; but, as the cupellation proceeds, it is from time to time cut down so as to keep the channel constantly at the level of the metallic bath. The litharge thus flowing from the apparatus accumulates on the floor of the smelting house, where it solidifies, and from whence it is removed as may be found convenient.

Before commencing a cupellation, it is necessary to prepare the cupel; and for this purpose, after lifting the iron dome, the old cupel, which has become thoroughly impregnated with litharge, is broken up and removed, in order that it may be passed through the blast furnace. The brick bottom of the apparatus is now moistened with water, and successive layers of finely-ground marl are well beaten in whilst in a somewhat damp state. The iron covering is replaced when the cupel has become sufficiently dry, and all the joints are well secured by luting them with clay.

The furnace is now charged with 160 quintals (about eight tons) of lead, which, to prevent injury to the bottom, is laid on a bed of straw; the fire is lighted, and the metal rapidly begins to melt. As soon as the fusion of the discs is completed, the bellows are slowly set in motion, and the surface of the bath becomes covered with a dark-coloured powder, consisting of oxide of lead, associated with various impurities. These pulverulent matters, the first of which are known as *Abzugs*, and those which follow as *Abstrichs*, do not enter into fusion;

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* When good marl for this purpose cannot be obtained, a mixture of clay and lime, or clay and wood ashes, is employed.

but the refiner now and then throws a shovelful of coal-dust on the surface of the bath, and, by the aid of a billet of wood, fixed crosswise on the end of an iron bar, draws the impure oxides towards the hole through which the litharge escapes, and finally on to the floor of the works. After the expiration of a short time, fused litharge begins to make its appearance; but that at first formed, being impure, from the presence of other oxides, is usually laid aside, and not mixed with the purer descriptions which soon follow; these are generally sold for glass-making and other purposes, in preference to being again reduced to the metallic state. The litharge produced during the last period of the cupellation invariably contains a considerable amount of silver, and, after being reduced to the metallic state, forms a portion of the charge worked in a subsequent operation.

The blast is now slightly increased, and the oxidation proceeds rapidly; small flaps, or valves, being frequently fitted to the ends of the nozzles, for the purpose of checking its strength, and distributing it more evenly over the surface of the fused metal. The operation is continued in this way until almost the whole of the lead has been converted into oxide; and the silver, retaining only traces of that metal, remains in the cupel, in the form of a metallic cake.

At the moment the oxidation of lead entirely ceases, a phenomenon known as the *brightening* takes place, and the operation is then known to be terminated. During the whole period of cupellation, the metallic bath is observed to present a more brilliant appearance than the brickwork of the furnace itself, and its temperature is in reality greater; since it not only participates in the heat developed by the combustion of the fuel employed, but also absorbs nearly the whole of that resulting from the oxidation of the portion of lead which is being constantly converted into litharge. As soon as the oxidation of the lead ceases, this source of heat is at once withdrawn, and the temperature of the bath of metallic silver rapidly descends to that of the brickwork of the furnace; losing at the same time the brilliancy of appearance which it before possessed.

On the other hand, at the instant the last portions of lead become oxidised, an extremely thin pellicle of litharge remains on the surface of the metal, which, rapidly decreasing, exhibits a series of brilliant prismatic colours. This exceedingly thin film of fused litharge finally disappears, leaving the surface of the metal uncovered. These successive rapidly-changing phenomena are called by the Germans, the *Blicken*, and by the French, the *Éclair*.

As soon as the operation is thus perceived to have terminated, the refiner throws water into the hearth, and removes the solidified cake of silver, which usually still retains a sufficient amount of lead to render its further purification necessary, by a process to be shortly described.

In some establishments, however, where the marl of which the cupel is composed is of good quality, the process of refining is conducted in the same apparatus. For this purpose the blast is continued during fifteen or twenty minutes after the appearance of the phenomena above described; and some small billets of wood, which are introduced through the litharge door, are kept burning over the surface of the metal. The plate thus obtained contains about 950 thousandths of silver.

The amount of silver resulting from each operation manifestly depends on the weight and richness of the lead operated on; and at Kremnitz a charge, consisting of about four or four and a half tons of auriferous silver lead, formerly produced from 1,900 to 2,800 oz. of silver, containing a certain proportion of gold.

At Clausthal, the treatment of 160 centners of ordinary lead, equivalent to about eight English tons, affords 56 marks of silver, about 450 ounces; 118 quintals of litharge, 21 quintals of cupel bottoms, and 15 quintals of abstrichs. The lead obtained in the same establishment from the fusion of roasted matts is more argentiferous than that directly produced from ores, and yields about 62 marks of silver per charge of 160 quintals.

The Continental cupelling furnace is not always provided with a chimney, but is usually placed under a large hood of masonry, which serves to carry off the smoke escaping from the charging hole and other apertures. Each operation, including the time necessary for preparing the cupel, occupies about thirty hours, and is attended with a loss of lead amounting to from 10 to 12 per cent. of the weight of metal subjected to cupellation. In some localities, coal is the fuel employed in furnaces of this description; but wood is much preferred, when it is to be obtained at a moderate price, since it affords a clearer and more oxidising flame, and can be managed with greater facility. At Clausthal, where wood is employed, the diameter of the furnace is 10 feet; the fireplace, 6 feet 6 inches in length, and 1 foot 6 inches in width. The metallic dome has, in that establishment, a diameter of 9 feet 10 inches.

Refining.—The purification of the silver obtained by the process above described is, on the Continent, frequently effected in a small reverberatory furnace, of which the bottom is composed of bone-ash, tightly rammed, whilst in a damp state, into an iron ring, and afterwards so hollowed out as to contain the bath of fused metal. The cupel, which must be thoroughly dry, and ought therefore to be prepared some time beforehand, is, whilst the furnace is still cold, so supported on bricks against abutments prepared for that purpose, as to form the bottom of the apparatus. It is charged with about one cwt. of the impure silver to be operated on, and the firing is continued until a bright red heat has been attained; the silver, which has by this time become fused, being exposed to the oxidising influences of the flame. In this way the lead contained in the alloy becomes oxidised, and the resulting litharge is absorbed by the cupel, of which the temperature is sustained until the oxidation of all but the last traces of lead has been produced. When this point has been attained, the surface of the fused metal becomes exceedingly brilliant, and reflects, as in a mirror, all the irregularities of the interior of the crown.

The bottom of the cupel is now pierced by a pointed iron bar, and the silver is run out into moulds, previously heated on a ledge of the furnace. In order to prevent the spirting, or "vegetation," of the bars, they are covered, whilst cooling, by a piece of dry wood, kept down by a weight; and in case of any irregularities making their appearance on the surface of the ingots, they are subsequently removed by hammering. This operation altogether occupies from four to five hours, and the resulting bars usually contain from 997 to 998 thousandths of silver. The actual loss of silver is almost inappreciable; but the diminution in weight experienced, on the crude metal from the cupelling furnace, is from 2½ to 5 per cent.

English System of treating Argentiferous Lead.—The lead obtained by the different processes before described, in addition to silver, contains various impurities, such as tin, copper, and antimony, which, when the metal is subjected to direct cupellation, are removed in the form of "abzugs" and "abstrichs;" but, when the previous concentration of the silver by crystallisation is resorted to, they materially interfere with the operation, and require to be removed by the process of calcination.

Calcination.—This consists in keeping the fused lead, exposed at a cherry-red heat to the oxidising influences of the gases passing through a reverberatory furnace, in which the calcination is effected. By this treatment the antimony, copper, and other impurities become oxidised, and, rising to the surface, are skimmed off, and removed by means of an iron rake. The length of time necessary for the purification of hard lead obviously depends on the nature and amount of the impurities with which it is associated; and, consequently, some varieties will be sufficiently softened at the expiration of twelve hours, whilst in other instances it becomes necessary to continue the operation during several days. The time necessary for sufficiently softening the argentiferous lead obtained from the Castillian furnace, when working ordinary ores, is about thirty-six hours.

The hearth of the furnace in which this operation is most frequently conducted consists of a large cast iron pan, about 1¼ inch in thickness, which may be 10 feet in length, 5 feet 6 inches in width, and 10 inches in depth. The fireplace, which is 20 inches in width, has a length equal to the width of the pan, from which it is separated by a bridge 2 feet in width. The height of the arch, at the bridge end, is 16 inches above the edge of the pan, whilst at the other extremity it is only 8 inches. All the angles of the casting are rounded in order to prevent breakage from expansion, and the softened lead is, when required, drawn off into a float by means of a hole bored in the bottom, near its outside edge. This, when closed, is stopped by means of a turned iron plug kept in its place by a weighted lever.

The charge of this furnace, which is about 11 tons, is first fused in a large iron pot, set in brickwork at the side, and is subsequently ladled into it through a sheet iron gutter prepared for that purpose. The amount of coals required for the calcination of a ton of ordinary hard lead is generally somewhat less than 3 cwt., and the cost of wages usually amounts to about 1*s.* per ton. The softened lead is cast into pigs, and is in that form taken to the crystallising pots.

In some smelting works the hard lead is calcined in an ordinary furnace of the reverberatory description, into which a slag bottom has been run. This usually rests on a layer of brickwork, supported on old railway bars, in such a way that, in case of any leakage taking place, the lead flowing through may be readily collected from the chamber formed beneath the hearth. When this furnace is employed, the calcination takes place at a higher temperature than in the iron pan, and consequently the operation only occupies one-fifth the time required in the latter; the consumption of fuel is about the same, but the loss of lead probably greater.

Concentration of Silver by Crystallisation—Pattinson's Process.—This process is founded on the circumstance, first noticed in the year 1829, by the late H. L. Pattinson, of Newcastle-on-Tyne, that when lead containing silver is melted in considerable quantities in suitable vessels, allowed slowly to cool, and at the same time kept constantly stirred, at a temperature near the melting point of lead, metallic crystals begin to form. These sink to the bottom, and, on being removed, are found to contain much less silver than the lead originally operated on. The still fluid portion, from which the crystals have been thus removed, will at the same time be found to be proportionately enriched.

This operation is usually conducted in a set of from 9 to 12 pots, which, if worked by hand, each contain 6 tons of metal; or, if cranes be employed, are 5 feet 4 inches wide and 2 feet 6 inches deep, and contain 10 tons of argentiferous lead. Each of these pots is provided with a separate fireplace, the heat from which is made to pass around it by means of a wheel flue, which can be closed at pleasure by a damper; the products of combustion finally escape into a large arched flue parallel with the line of pots.

We will suppose that the lead under treatment contains about 20 oz. of silver per ton, and is introduced into pot No. 5, Fig. 69. This metal when fused is carefully skimmed with a perforated ladle, and the fire at once withdrawn. The cooling of the metal is now hastened by sprinkling water on its surface; and whilst the temperature is being thus lowered, it is kept constantly stirred with a chisel-pointed iron bar, called a slice. All those portions which become solidified, and adhere about the side of the pot, are also removed and forced under the surface of the metal, in order that they may again become melted. Under this treatment, crystals soon begin to make their appearance; and as they fall and accumulate at the bottom, they are removed by means of a large perforated ladle, in which, after being well shaken, they are first allowed to drain over the pot whence they have been taken, and afterwards carried over to the next pot (No. 6) to the left of the workmen. This operation is continued until about two-thirds of the lead originally present in pot No. 5 has been transferred in the form of crystals to pot No. 6, at which period the lead remaining in pot No. 5 will contain about 40 oz. of silver per ton, whilst that transferred to No. 6 yields only 10 oz. The rich lead in the bottom of No. 5 is now ladled into the pot No. 4 next on the right.

In this way a fresh supply of calcined lead is constantly introduced, the resulting crystals passing continually to the left of the feeding pot, whilst the enriched lead, remaining in the bottom, is ladled into the pot on the right. Each pot in succession, when it has become filled with metal of the proper produce for silver, is in its turn crystallised; the poor lead passing to the left, and that which has become enriched to the right in the series. It is evident that by this means the crystals obtained from the pots to the left of the feeding pot gradually become deprived of their silver, whilst the rich lead passing to the right is continually enriched. The final result, therefore, is, that at one end of the line of kettles the lead contains but little silver, whilst at the other extremity it becomes exceedingly argentiferous.

The desilverised or *market lead* obtained by this process should never contain above 12 dwt. of silver per ton, and is frequently much poorer, whilst the rich lead is sometimes so concentrated as to yield 600 oz. per ton. This rich lead is passed to the refining furnace for cupellation.

The ladle employed, when manual labour is made use of, is 16 inches in diameter, 5 inches in depth, and pierced with ½ inch holes. When cranes are employed, the ladles are 20 inches in diameter, 6½ inches in depth, and are pierced with holes ¾-inch diameter: thickness of iron, ½ inch; length of handle, 9 feet 4 inches. The large *baling ladles* used for turning back the bottoms, are 14 inches in diameter and 8 inches deep, and have a handle 7 feet long.

Two crystallisers are employed in working each pot, and one fireman every twelve hours is required for each set. By the use of cranes a ten-ton pot can be worked as quickly, and at the same expense for labour, as a six-ton pot by hand ladles.*

Fig. 69 represents a plan, and Fig. 70 an elevation, of a set of Pattinson's pots, fitted with cast iron cranes arranged according to the most approved system. Nos. 1, 2, 3, 4, 5, 6, 7, 8, and 9, are working pots, and No. 10 the market pot, out of which the desilverised lead is ladled into moulds; and which, from only receiving the crystals from No. 9, and not having a bottom of enriched lead left in it, has only two-thirds the capacity of the other pots.

A long ash-pit A extends the whole length of the set, and is partially covered by the iron platform B, supported on iron pillars. The fireplaces *a* are provided with iron doors.

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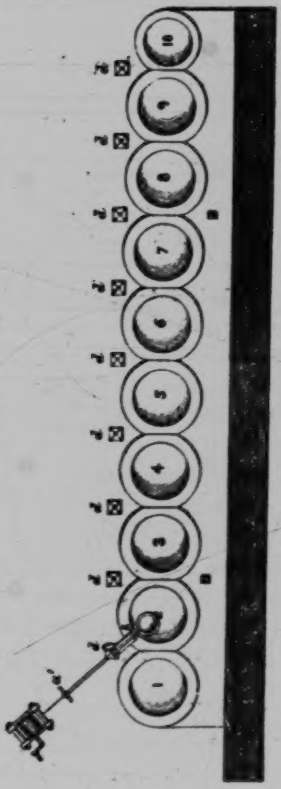
* With cranes, two men can work a ten-ton pot in about two hours.

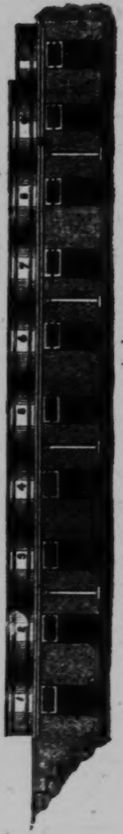
When, during the operation of taking out crystals, the perforated ladle becomes chilled, it is heated to the proper temperature by being dipped into the pot of hot lead into which they are turned over.

Fig. 69.



Plan of Crystallising Pots.

Fig. 70.



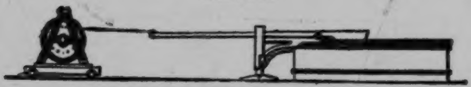
Elevation of Crystallising Pots.

Fig. 71 represents an elevation of one of the pots and the crane, winch, and chain, by which the ladle, when full of crystals, is withdrawn.

In order to work by the aid of this arrangement, the potman sinks the ladle sidewise to the bottom of the kettle, and having turned it over so as to be full of crystals, he attaches a hook to the cross handle *a'*, of the ladle, Fig. 69, which is then withdrawn by the other workman turning the winch.

In doing this, the iron shank slides over the roller *b*, at the end of the crane *d*; and as soon as it is withdrawn from the fused metal, the first workman, who guides the handle during the operation, slips it into one of the cheeks *c*, at the back of the crane, where it becomes firmly secured. The ladle, full of crystals, is thus suspended over the pot from which it has been withdrawn, and after being allowed a short time to drain, it receives a few shakes by jerking the iron handle. The crane is now swung round, the shank slipped out of the catch, and the crystals deposited in the next pot on the left. This is continued until the necessary amount of crystals has been withdrawn, when the rich lead remaining in the bottom is taken out, in the same way, by a ladle without perforations, and turned over in the next pot on the right. In some establishments the lead remaining in the bottom of the rich pot (No. 1) is further concentrated by allowing it to cool to the crystallising point, and then pressing it with the convex side of one of the large perforated ladles. The still liquid alloy is thus made to enter the bowl through the holes with which it is pierced, and is taken out with a smaller unperforated dipper. The lead thus obtained will evidently be richer than the crystals remaining in the kettle.

FIG. 71.



ELEVATION OF POT AND CRANE.

The following memoranda relative to the Wildberg Smelting Works afford some practical data respecting the costs incurred, and results obtained, in the desilverisation of argentiferous lead in six-ton pots by hand ladles:—

Crystallising from July 1858 to May 1860. Cost of coal, delivered, 40*s.* to 45*s.* per ton. Average assay of calcined lead, 28 oz. of silver per ton. Lead concentrated to 340 or 370 oz. of silver per ton. Assay of market lead, from 7 to 9 dwt. per ton. Number of pots employed in crystallising, 9. Market lead obtained, 30,032 Prussian centners. Rich lead cupelled, 2,486 Prussian centners. Weight of coals consumed, 10,807 ctr., or about 7 cwt. per ton of market lead.

Cost on one ton of market lead—

	Th.	sgr.	pf.	£.	s.	d.
Wages	3	7	3	0	9	9
Coals	5	3	10	0	16	4
				£1	6	1

When worked regularly so as to leave a bottom of one-third the weight of their total charge, the richness of each pot in a series will be found to be about double that of the kettle next it to the left. Assays of the contents of each of a set of nine pots, in which the calcined lead worked, contained 21 oz. 9 dwt. 8 gr. of silver per ton, gave the following results :—

	oz.	dwt.	gr.	Remarks.
No. 1	219	16	0	Bottom of No 1, or rich lead, gave 358 oz. 8 dwt. per ton.
„ 2	129	8	0	
„ 3	79	8	0	
„ 4	41	1	8	
„ 5	21	9	8	Lead fed into No. 5 pot.
„ 6	11	4	0	
„ 7	4	5	0	
„ 8	2	0	0	
„ 9	0	18	16	
Market Pot	0	9	0*	

French Process of Crystallisation.—Instead of effecting the crystallisation of argentiferous lead in a series of iron pots or kettles, and then removing the crystals formed by means of ladles, worked either entirely by hand or with the aid of cranes, a process has been recently introduced by which manual labour is almost completely dispensed with. The apparatus employed for this purpose essentially consists of two cast iron vessels, the first of which is called the melting pot, and the second the crystallising pot. The former has the same capacity as the latter, and is so set that its bottom is at a higher level than the top of the other. This melting pot is provided, on the side next the crystalliser, with a discharge pipe, which can be tightly closed by means of a slide valve.

The crystallising apparatus consists of an ordinary cast iron pot provided with a vertical stirrer. This pot has, at opposite sides of the bottom, two discharge pipes fitted with slide valves; and each heated by a small separate fire to prevent its becoming obstructed by the cooling of the metal. Beneath each of these pipes is placed a pot for the reception of the liquid alloy which, on opening the valves, drains from the crystals.

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* Cast into moulds and sent into the market as soft lead.

The stirring apparatus consists of two wrought iron shafts working vertically in the centre of the pot; of these one is solid, and supported on a step cast on to the pot, whilst the other consists of an iron tube enclosing the former. Each axle has, at its upper extremity, a mitre wheel, and, as the outer shaft is shorter than the inner one, another mitre wheel is made to work horizontally between them in such a way as to cause the two shafts to revolve in contrary directions. To the lower end of each is connected an iron stirrer, provided with knives, and almost touching the sides of the pot, so as to protect them from any incrustation of lead. The object of this stirrer, which is set in motion by a belt from a small steam-engine, is to promote throughout the mass the production of the uniform temperature necessary for the crystallisation of the metal; and at the same time to so compress the crystals formed, as to cause them to separate readily from the liquid alloy.

It is evident, that, with the formation of increasing quantities of crystals, the friction of the stirrers becomes greater; and when a certain amount of crystals has accumulated, the friction increases to such an extent that the power of the engine is insufficient to continue the motion of the shafts. For this reason the engine employed should not be too large; or an arrangement may be made for causing the strap to slip when the proper accumulation of crystals has taken place. These conditions may be so adjusted as to suit any system of crystallisation; but when the alloy is to be reduced to one-third the weight of the lead operated on, the apparatus must be brought up by the friction as soon as two-thirds of the lead have become crystallised. The pots are heated by a suitable fireplace, and above those into which the liquid alloy is run is a crane by which the enriched lead is lifted out, by means of iron eyes cast in the metal, and brought back to the melting pot for the purpose of being further enriched.

To use this apparatus, such a quantity of original lead must be melted as corresponds to the capacity of the crystallising pot, and as soon as it is fused it is run into it, and the operation of stirring commences. The formation of crystals is effected by lowering the temperature in the usual way, and, as soon as a sufficient amount has been formed, the lateral valves are opened, and the enriched alloy is run into the receiving pots.

Lead, of the same richness in silver as the crystals remaining in the crystalliser, is now melted in the melting pot, in sufficient quantity to make up, with them, another charge. This is tapped in upon the crystals, the mixture again crystallised, and a certain quantity of liquid alloy obtained as before. These operations are repeated, until a concentrated rich alloy, ready for cupellation, is produced, on the one hand; and, on the other, desilverised crystals, which, after being run into moulds, are ready for market.

This process can be continued with unbroken regularity, as soon as there is accumulated a sufficient amount of each class of lead to make, with the crystals remaining in the crystalliser, a full charge for that pot. We are not aware whether this process is found more economical than the ordinary one in which the crystals are removed by means of ladles and cranes.

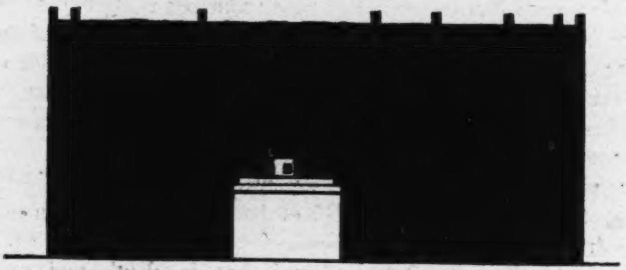
Refining.—The cupellation of argentiferous lead is in this country conducted on a hearth composed of bone-ash, forming the movable bottom of a reverberatory furnace. The cupel or *test* is contained in an oval iron ring, usually six inches in depth, and generally about four feet in its greater, and three feet in its lesser diameter. In order to better support the bottom of the test, the frame is provided with cross-bars, 4½ inches wide, and half an inch in thickness. This framing, or *test ring*, is most frequently made of wrought iron bars held together by rivets, but in some cases it is formed of cast iron, and is then, including the bars across its bottom, cast in one piece. In order to prepare a test, the frame is filled with bone-ash, well beaten in layers, after having been slightly moistened with water, containing a small quantity of pearlash in solution; the presence of a minute proportion of that salt having the effect of giving consistency to the cupel when heated.

After the framing has been thus filled with moistened bone-ash, solidly beaten down, the centre is carefully scooped out with a small trowel, until the sides are left 2 inches in width at top, and 3 inches at bottom, whilst the thickness of the sole itself is about 1½ inch. At the front or wide end of the test, the bone-ash border is allowed to stand for a width of six inches, and an opening is cut through the bottom; which, by communicating by means of a channel with the fluid litharge in the annular cavity, formed between the cupel and the edge of the metallic bath, allows the fused oxide of lead to run off as fast as it is produced.

The test, when thus prepared, is kept several days to become thoroughly dry, and when required for use is placed in the refining furnace, of which it forms the bottom. Figs. 72, 73, and 74 represent respectively an elevation, a horizontal and a vertical section, of the refinery most commonly employed.

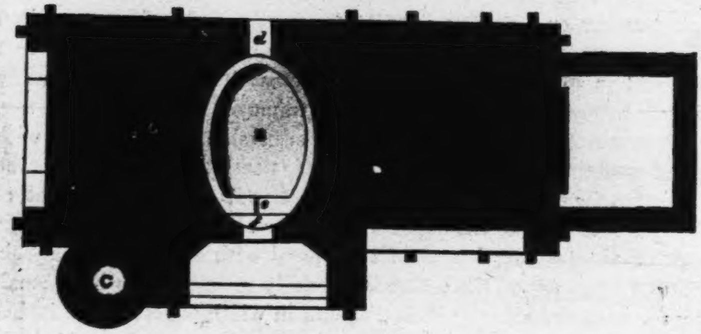
The size of the fireplace A varies with the other dimensions of the apparatus, but is usually nearly square, and in a furnace of ordinary size, may be about 2 feet by 2 feet 4 inches. This is separated from the body of the furnace, by a bridge of from 14 to 18 inches in width, so that the products of combustion pass directly over the surface of the cupel, and from thence escape by two separate apertures, into the main flue.

FIG. 72.



ELEVATION OF REFINERY.

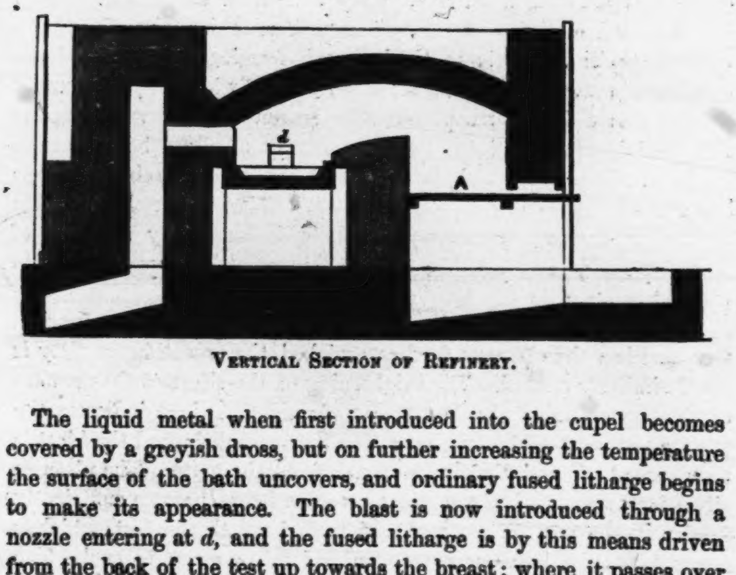
FIG. 73.



HORIZONTAL SECTION OF REFINERY.

The test B, when sufficiently dried, is placed in the refinery, of which, as before stated, it forms the bottom; being closely jammed by means of the wedges *a*, of which there are four, resting on two iron bars, against an iron ring *b*, firmly built into the masonry of the furnace. When first lighted, it is necessary to regulate the fire with great care, since, if the cupel were too suddenly heated, it would be liable to crack and exfoliate. As soon, however, as it has become properly annealed, it is heated to incipient redness, and its cavity nearly filled with the rich lead, to be operated on. This has been previously fused in an iron pot c, provided with a fireplace and a gutter *e*, through which the melted lead is ladled into the cavity of the test.

FIG. 74.



VERTICAL SECTION OF REFINERY.

The liquid metal when first introduced into the cupel becomes covered by a greyish dross, but on further increasing the temperature the surface of the bath uncovers, and ordinary fused litharge begins to make its appearance. The blast is now introduced through a nozzle entering at *d*, and the fused litharge is by this means driven from the back of the test up towards the breast; where it passes over the *gate*, *e*, and, falling through the aperture *f*, between the bone-ash and the iron frame of the cupel, is received in a small shallow cast iron pot, running on wheels, and provided with a long handle.

Some refiners, instead of feeding the test with rich lead in a fused state, prefer to introduce it into the furnace in the form of pigs. In this case the apparatus is provided with one or more iron-lined openings *g*, called pig holes, through which the pigs of metal are introduced. The blast, which is usually supplied by means of a fan, thus not only sweeps the oxide along the surface of the fused metal towards the breast, but also supplies the oxygen necessary for the formation of fresh litharge.

In proportion as the surface of the metallic bath becomes depressed through its constant oxidation and the continual removal of the resulting litharge, additional metal is supplied from the melting pot C, so as to raise it to its former level; and in this manner the operation is continued until the lead in the bottom of the test has become so much enriched as to render it necessary that it should be tapped. When the lead obtained from the rich pot affords about 600 oz. of silver per ton, this operation is usually performed every eight hours, during which time 32 cwt. of metal have been introduced into the cupel, and six cwt. of enriched alloy remain in the metallic state, The removal of the concentrated argentiferous lead is effected by carefully drilling a hole in the bone-ash forming the bottom of the cupel, and running it off into a pot on wheels, which is placed under the test frame for its reception. The reason for thus removing the rich lead is to prevent too large a quantity of silver being carried off in the litharge; which is the case when lead containing a large amount of that metal is operated on. When the rich lead has been thus drawn off, the tapping hole is closed by a pellet of wet bone-ash, and another charge immediately introduced into the test.

As soon as the whole of the rich lead has been subjected to cupellation, and has thus become further enriched, the argentiferous alloy is itself similarly treated. The brightening of the silver at the moment of the separation of the last traces of oxidisable lead indicates the precise period at which the operation is terminated; and the blast is then turned off and the fire removed from the grate. The plate of silver is now allowed to set, and as soon as it has become sufficiently hardened, the wedges are removed from beneath the test, which is lowered upon a small iron waggon and taken away to cool. The silver is subsequently detached; and any adhering particles of bone-ash or litharge removed by scraping with a wire brush.

A refinery of the usual dimensions will work off 4 pigs (4 cwt.) of lead per hour, and consumes about 6 cwt. of coal per ton of lead operated on.

The expenses, in this country, of refining one ton of rich lead containing 600 oz. of silver, in a locality where coal costs 10s. per ton, are as follow:—

Refiner's wages	4*s.*	6·0*d.*
Coal, 6 cwt.	3	0·0
Engine, wages	1	9·0
„ coal, 3·1 cwt.	1	6·6
Pearlash	0	3·5
Bone-ash, 17·5 lbs.	3	3·4
Repairs	0	7·0
Total	14*s.*	11·5*d.*

The plate thus obtained usually contains from 997 to 998 thousandths of silver.

The following statement gives the fuel consumed, and results obtained, in 1858, at a large English establishment for the treatment of argentiferous lead.

Cost of crystallising 2,838 tons of soft lead and 2,019 tons of hard lead containing 114,738 oz. of silver, including expenses of improving, reducing, and refining. The pots employed each contained 10 tons of lead, and were worked by means of cranes :—

Average assay	23 oz. 13 dwt. 3 gr. per ton.
Average allowance*	7 „ 10 „ 2 „ value, £2 1*s.* 3*d.*

Coal consumed 7 cwt. 16 lbs. (at 14*s.* per ton) per ton of lead.

Crystallisers paid per piece 1*s.* 3*d.* each per pot of 10 tons. Refiners paid 30*s.* per week. Reducers 4*s.* per shift, to run 60 pigs from pot drpss, 40 from litharge, or 25 from hard dross. Improvers 21*s.* per week of 6 days or 5 nights.

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* The allowance made by the seller of argentiferous lead to the buyer, for the purpose of covering the expenses of desilverisation, is calculated according to what is called the Newcastle Scale, which is as follows :—

Newcastle Scale of Deductions on Desilverising Lead.

Lead under and not exceeding	Deducted.
10 oz. per ton	5½ oz.
20 „ „	6½ „
30 „ „	7½ „
40 „ „	8 „
50 „ „	9 „
60 „ „	9½ „
80 „ „	10 „
100 „ „	11½ „

In calculating the quantity of silver in lead, no fraction less than $\frac{1}{10}$th of an ounce per ton is reckoned. The surplus over the above allowance is paid for at the London price of the day, less ¼*d.* (one farthing) per oz. for carriage and other charges.

	Total cost.			Cost per ton.		
	£.	s.	d.	£.	s.	d.
Potmen's wages	2,227	14	9	0	9	2
Labourers at pots	115	15	1	0	0	6
Labourers	167	14	9	0	0	8½
Coal and coke, crystallising	895	11	4	0	3	4
Reducing	260	9	0	0	1	1
Improving	300	9	9	0	1	3
Refining	366	6	5	0	1	7½
Landing and delivery	269	13	10	0	1	1½
Castillian furnace	121	5	10	0	0	6
Tools, Repairs, and Materials	650	9	0	0	2	9
Salaries	378	11	0	0	1	7½
Rent, Taxes, and Furnaces	441	19	6	0	1	10
Gas	88	19	6	0	0	4½
Sundries	53	5	3	0	0	3
	£6,338	5	0	£1	6	1½
Loss of lead*	80	0	0	0	0	4
Silver in market lead	1,007	0	0	0	4	2
	£7,425	5	0	£1	10	7½

LIQUATION.—The process of extracting silver from argentiferous copper by liquation is still employed in some localities, although it has been now generally supplanted by the cheaper and more efficient systems of treatment by the wet way. In the Upper Hartz, however, it may yet be seen in full operation, as the Ziervogel and other wet processes, as well as the ordinary methods of treatment by barrel amalgamation, do not appear to be applicable to the complex ores, of that district, containing lead and silver.

This process is based on the circumstance that when copper containing silver is alloyed with a certain proportion of lead, and afterwards heated above the melting point of the latter metal, but below the temperature at which copper enters into fusion, the lead becomes liquid and drains or sweats out, carrying with it the greater portion of the silver, whilst impure copper remains behind in a cavernous or spongy form.

It has been found by experience that, to conduct this operation successfully, the lead should be present in the alloy formed, in the proportion of eleven parts of that metal to three of copper, and that for every part of silver present about five hundred parts of lead should be employed. In the Hartz, according to Lamborn, black copper containing one-sixth per cent. of silver can be treated with advantage by liquation; but when only one-eighth per cent. of this metal is present, its extraction does not pay the expenses incident to the operation.

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* The actual loss of lead is greater than that shown, but there was the usual trade allowance of 14 lbs. on every ton of silver lead weighed into the factory, which thus reduced the deficit to that above stated.

This process is not however generally applicable to the extraction of silver from ores in which that metal forms the most valuable and important constituent; and since it has been often described with much detail in various German and other treatises on metallurgy, we refer those who desire information on this subject to the authors in question, rather than attempt to describe an operation of which we have had no practical experience, and relative to which we possess no special information. The process of liquation will be found described in the works of Villefosse, Kerl, and others.

CHAPTER XXII.

TREATMENT OF ARGENTIFEROUS GALENA AT PONTGIBAUD.

PREPARATION OF LITS DE GRILLAGE—ROASTING—PREPARATION OF LITS DE FUSION—SMELTING IN CASTILLIAN FURNACE—IMPROVING OR CALCINING—CRYSTALLISING—REFINING—REDUCING—RE-SMELTING RICH SLAGS—ROASTING MATTS—TREATMENT OF CALCINED DROSS—TREATMENT OF LEAD CINDER—TREATMENT OF LEAD FUME—LOSSES OF LEAD AND SILVER—SUMMARY OF COSTS.

THE method of smelting employed at Pontgibaud affords an example of the treatment of highly silicious lead ores rich in silver.* These ores occur in large veins of quartz and feldspar traversing gneiss. The average produce of the ore, as extracted, scarcely exceeds 6 per cent. of lead, and consequently very large quantities must be passed through the different washing processes in order to obtain the amount of mineral, averaging about 50 per cent., which is annually smelted in this establishment. As much as possible, however, of what is called *massif* or cobbed ore is carefully selected, in order to avoid exposing it to unnecessary mechanical loss. Recently the proportion of massif to washed ore has become much greater than formerly, particularly since the discovery of a new mine yielding ores very rich in silver. The effect of this has been to render the ores more refractory, since the assay of hand-picked ores for lead is seldom higher than 40 per cent., whilst the proportion of silicious gangue is much greater than in the washed ones.

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* The gangues of the Pontgibaud ores are similar in composition to those of many of the silver-bearing veins of Mexico, Nevada, and other parts of the American Continent. We shall consequently describe, with considerable detail, the various operations conducted in that establishment, since the system there employed is well adapted for the treatment of argentiferous minerals containing a large amount of silica, wherever fuel and lead ores can be obtained at reasonable prices.

The mines of Pontgibaud, Puy-de-Dôme, France, have at various periods afforded large quantities of argentiferous galena, but were never so productive as at the present time. The re-working of the concessions was commenced by a local company in 1825, and the operations were continued under the management of French engineers until the year 1852, when the property was transferred to an Anglo-French Association, under the management of Messrs. John Taylor & Sons, of London. We were at that period for some time occupied in re-modelling the smelting works, and introducing into them various modern appliances, but great improvements have been since made, and more particularly in the apparatus employed for roasting the ores and their preparation for the blast furnace.

The various ameliorations introduced into the system of treatment at Pontgibaud are, to a great extent, due to Mr. W. Hutchison, the present manager of the Smelting Works, to whom we are indebted for the drawings of the different furnaces, and for very copious and admirably arranged notes, affording the practical results of the several operations as now conducted.

It is to the uniformly silicious nature of the Pontgibaud ores, together with the impurities contained in the work lead obtained, that many of the difficulties experienced in smelting them are due, and which have rendered necessary the adoption of the special processes employed at this establishment. These difficulties are further increased, particularly in the matter of cost, by the remoteness of the locality in which the mines are situated; Pontgibaud being at a great distance from any centre of industry from which fuel and fluxes can be obtained at moderate prices.

All the ores are delivered at the smelting works in a state of fine powder; that is to say, the coarsest will pass through a sieve with apertures of 4 m.m. in diameter. As many as seven varieties are received monthly from the different mines belonging to the company. They vary considerably in richness, both as regards lead and silver, and the amount of gangue which they contain; this gangue is *always silicious*, although generally associated with small quantities of sulphate of baryta, arsenical pyrites, iron pyrites, blende, &c.

The produce of each mine is sampled on the 1st of each month, and immediately delivered to the smelting works; where the operations are so conducted that all the ores received during one month may be converted into pig lead before the next month's deliveries take place. This system, besides its convenience, has the advantage of enabling the smelter to check the calculated produce by the actual monthly returns. Each sample of ore is tried by two assayers, one on the part of the mines and the other on that of the smelting works; if their results differ to the extent of 1 per cent. each repeats his assays. In case of a constant difference, a sample is sent to a professional assayer, and the result he obtains is considered final; but this course is seldom resorted to.

All the assays are made in an iron crucible, and when properly conducted yield results quite as high as those obtained by the humid way; a circumstance probably owing to the impurities in the lead button compensating for the slight loss by volatilisation.

It will be necessary to bear this in mind, in comparing the losses of lead at the Pontgibaud works, with those experienced in other establishments where ores of a different nature are treated, and where the assays are, generally speaking, less carefully executed.

In the treatment of these ores nine distinct operations are necessary, viz.:—

1.—Preparation of *Lits de Grillage*.
2.—Roasting.
3.—Preparation of *Lits de Fusion*.
4.—Smelting in Castillian blast furnace.
5.—Improving.
6.—Crystallising.
7.—Refining.
8.—Reducing.
9.—Re-smelting rich slags.

In addition to the above, the routine of the establishment renders four supplementary operations necessary, viz.:—

a.—Roasting matt.
b.—Treatment of calcined dross.
c.—Treatment of lead cinder.
d.—Treatment of lead fume.

1. *Preparation of Lits de Grillage.*—Although the ores do not differ materially as regards the nature of their gangues, they vary considerably in richness; and consequently also in the proportion of earthy matters present.

It has, therefore, been found important, before commencing their treatment, to prepare a uniform mixture of the whole sampling.

On this depends the regularity of the subsequent operations, and, in a great measure, their economical working.

As it is impossible to thoroughly mix the whole weight of the various parcels of ore (often amounting to upwards of 300 tons), a *lit* of twenty tons, or a little more than the quantity usually roasted per diem, is prepared by weighing out, and spreading in thin layers, one above another, the exact proportion of twenty tons, which each parcel bears to the total weight delivered.*

The "lit" being finished, the charges are made by successively cutting down with a shovel the pile of stratified ore; and in such a way that every ton of the mixture removed shall have nearly the same composition as the entire mass.

Experience has proved the advantage of this arrangement over that of charging the hoppers direct from the several heaps in the ore magazine.

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* Tons of 1,000 kilos.

The following will serve to show the kind of ore comprised in an ordinary monthly sampling:—

	Dry ore. kilos.	ASSAYS.	
		Lead per cent.	Silver per M. kilos.*
A.—Washed ore	134,504	56½	1·100 grammes.
B.—Massif	37,520	40	0·725 „
C.—Washed ore	21,960	47½	1·137 „
D.—Washed „	9,393	66¾	1·650 „
E.—Massif	41,378	47	2·300 „
F.—Washed ore	29,515	49	1·206 „
G.—Washed „	26,528	66¾	0·380 „
Total	300,798		

To this mixture of ores are added the matts resulting from the previous month's smelting in the blast furnace, which are, after being ground and roasted *dead* in a reverberatory furnace, treated exactly as ordinary ores.

The total weight of matt mixed with the ores in the above instance was 35 tons, containing 14 per cent. of lead, and 400 grammes of silver per ton of 1,000 kilos.

For preparing a lit de grillage of 20 tons, the proportion required of each of the above parcels (wet weight) was as follows:—

Ore A.—8,094 kilos.
„ B.—2,174 „
„ C.—1,374 „
„ D.— 570 „

Ore E.—2,454 kilos.
„ F.—1,780 „
„ G.—1,594 „
Matt.—1,960 „

The average amount of moisture contained in these ores was 6 per cent.

It will be observed that the proportion of roasted or calcined matt in the above mixture is about 10 per cent.; but this proportion constantly varies from month to month. The object of adding it to the lits de grillage is, that its oxide of iron may serve as a flux in the subsequent operation.

In some cases, *i.e.* when the ores are more than usually quartzose, 10 per cent. of matt is found to be insufficient for this purpose, and ground *scories de forge*, or mill cinders, are added as a substitute: not unfrequently the previous month's production of matt does not amount to 10 per cent., and then also iron slags are employed. As a rule, 10 per cent. of matt, or 15 per cent. of mill cinder, is sufficient, but it often happens that these proportions must be considerably exceeded.

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* The average produce of these ores for silver will be found to be about 39 oz. per English ton.

It would perhaps appear preferable to add iron slag directly to the mixture prepared for the blast furnace, as is done in some other works; but repeated experiments have led to the adoption of the method now employed; since its value as a flux in roasting, by the Pontgibaud system, is almost as great as its subsequent utility during the operation of smelting.

Two men are employed in the preparation of the roasting mixture; they are paid two francs each per day, and in that time prepare the mixtures, and charge each furnace with eight tons of ore. The cost of this operation is 0·25f. (2½*d.*) per ton.

2. *Roasting.*—The ores are roasted in very large reverberatory furnaces worked from both sides. There are three of these at Pontgibaud, but two only are in constant operation; all have exactly the same form and dimensions; their great width, as well as their great length, is a most important feature as regards economy of fuel.

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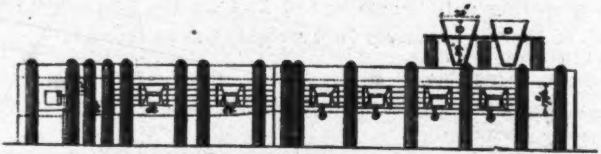
Fig. 75.

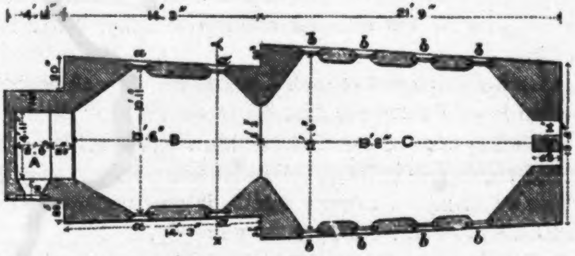


Roasting Furnace.
(*Elevation.*)

Fig. 75 represents an elevation, Fig. 76 a horizontal section through the working doors, and Fig. 77 a vertical section, on the line X Y, of the roasting furnace. The exterior is built of cut lava, and the sides and roof of firebrick; the sole is laid with hard common bricks. The outside is plated with iron between the bottoms and tops of the doors *a*, *b*, as shown by the horizontal lines. The doors *a* nearest the fireplace are a little smaller than the others. The fireplace A is divided from the fusing hearth B, by a firebrick bridge 20 inches in width; the ore being admitted into the hearth C, through the hoppers D. The bottom E, beneath the lining of common bricks, is composed of a mixture of sand and slag well beaten in. The tap hole is shown at *c*, Fig. 77. This apparatus requires but little repair.

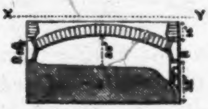
When working, this furnace contains six tons of dry ore and matt, divided into three charges of two tons each, severally occupying about one-third the surface of the entire sole. There are six doors on either side, two of which correspond with one or other of the charges, and enable the workmen to turn or advance the ore when required.

FIG. 76.



ROASTING FURNACE.
(*Horizontal Section.*)

FIG. 77.



ROASTING FURNACE.
(*Vertical Section.*)

The different parts of the furnace occupied in succession by each charge may be distinguished as follow:—

1st.—Drying bed, immediately under the hoppers.
2nd.—Desulphurising or oxidising bed, widest part of furnace.
3rd.—Agglomerating bed, next to fireplace.*

The first two are on the same level, the third about 20 c.m. lower than the others.

The peculiar form of the agglomerating or fusing bed has for its object the equalisation of the temperature over that portion of the furnace in which the greatest heat is required.

The ore arriving on this bed is fused and run out in the form of a liquid slag. At intervals of six hours a charge of melted ore is withdrawn, and the other charges in the furnace are advanced a stage; whilst a fresh charge is let down through the hoppers upon the drying bed. The time each charge remains in the furnace is consequently 18 hours. Eight tons of ore and matt are thus roasted in each furnace in the course of 24 hours, with a consumption of about 2,000 kilos. of coal, and 6 per cent. of lime. The consumption of iron slags averages 7 per cent.; but, whatever the proportions of this flux, or of lime, may be, the quantity of ore charged remains constant—that is to say, 1,800 kilos.

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* The agglomeration of the ores at Pontgibaud is in reality a fusion, without reduction.

Four men are employed at each furnace, per shift of twelve hours; the foreman is paid 2·20f. per day, the others 2f.

The general system of roasting in this furnace will be understood from the foregoing description; but, in order that it may be fully comprehended in all its various details, it may perhaps be necessary to explain the mode of working more minutely.

Let us, for this purpose, suppose that the usual regularity of the operations has been uninterrupted during the night, and that at six in the morning we accompany the men to their work. We shall find the bed next the grate empty, the charge having, about two hours previously, been run out upon the floor, on one side of the furnace. The charge in the middle bed having been roasted dead, is heaped up in the throat ready for being advanced into the agglomerating bed, which is still at a red heat, although the fire has been allowed to burn down in the grate. The foreman begins by throwing a shovelful of coals on the fire to create a blaze and light up the interior, so that the men may see their work. All hands then commence busily advancing the ore with long paddles. Two men advance it upon the agglomerating bed, and pile it up as near the bridge as they can; whilst two others push forward the ore from the drying bed into the middle or desulphurising hearth. In the latter, the ore is spread evenly over the sole, and two or three shovelfuls of slaked lime are afterwards scattered over its surface; especially on the side next the fire, to prevent the formation of a crust of partially-agglomerated ore very difficult to calcine.

The time required for advancing the two charges is about forty minutes. A new charge is now let down from the hoppers D, and spread over the drying bed with a rake. The doors are then all closed, the grate charged with coals, and the fire increased until the furnace has acquired a bright red heat.

In about three quarters of an hour the ore in the middle bed has become sufficiently hot to scintillate when stirred, and to give off sulphurous acid vapours. Two men, one on each side, now commence turning over the charge with paddles, which they do repeatedly, first forwards towards the fire, and then backwards towards the flue; taking care that the whole of the ore is turned, and that fresh portions are constantly exposed to the action of the heated air. From time to time a shovelful of slaked lime is thrown in and worked up with the ore to keep it "dry;" especially on the side next the fire, where it is most liable to clot. The operation of turning is continued without interruption until the charge in the agglomerating bed is ready to tap, when, during the time of tapping, &c. there is an interval of nearly half an hour; but it is afterwards renewed and continued until the expiration of six hours, or at least until the ore has been roasted dead. When the mixture of ores is good, *i. e.* when it is moderately rich in lead, and contains at the same time a good deal of oxide of iron, the charge in the agglomerating bed requires but little working. It then melts easily, and in about two hours and a half after the firing commences it is ready to draw. But, should the mixture contain an excess of silica, it is difficult to get the ore into a liquid state. Part of the charge attaches itself to the sole, and another portion is drawn out, after great trouble, in a tough pasty condition. A great loss of lead, and considerable waste of fuel, are the consequences. For this reason it is advantageous to add oxide of iron to the ore, and the cheapest way of doing this, at Pontgibaud, is in the form of iron slags, although their cost is 27f. per ton.

Assuming the charge worked to be easily fusible, it is only requisite to paddle it once towards the bridge; about three-quarters of an hour after beginning to fire. It is also necessary, just before tapping, to rake over the bottom, to be certain that none of the ore is sticking to it, and that the whole charge is perfectly fluid. When this is found to be the case, the tap hole is unstopped, and the charge run out upon the floor; where it is prevented from spreading beyond certain limits, by a small dam or ridge of ore and sweepings, &c. reserved for that purpose. No lead is reduced in this process, but a certain quantity of very rich sulphide of lead is generally met with at the bottom of the charge. This is particularly the case when the ore is rather rich in lead, and has been imperfectly roasted.

The bulk of the charge of roasted ore is composed of a clean black slag containing from 30 to 40 per cent. of lead. The whole of the sulphate of lead formed in the middle bed is subsequently decomposed by the silicic acid present. As soon as the charge has been run out, the tap hole is examined, and any adhering ore is cut away with a long chisel; after which it is again stopped. The bridge and sides are then looked at to ascertain whether they require repairs; since holes are often eaten out by the melted ore, which require to be stopped with clay after almost every charge. The fire is now suffered to burn down, to allow of clearing the grate of clinkers. The damper is at the same time partially closed, and preparation is made for the advance of another charge, when the time for doing so shall have arrived. Charge thus succeeds charge at regular intervals of six hours: the loss of weight in roasting amounts to about 10 per cent. of the quantity charged.

The loss of metal by volatilisation is extremely difficult to estimate, as it is scarcely possible to obtain a fair sample of the roasted ores. Several experiments, undertaken with this object, have fixed the loss of lead at from 2 to 3 per cent. of the total quantity present. The loss of silver is still more difficult to determine, and no result which can be depended on has yet been obtained.

The cost of roasting ore per ton is as follows (the dry ore per charge being 1,800 kilos.):—

Coal,	280 kilos. at 35 f.	9·80 f.
Lime,	60 „ at 24 f.	1·45
Scories de forge,	70 „ at 27 f.	1·90
Labour		2·90
Tools and repairs		0·80
Sundries (including superintendence)		2·90
		F. 19·75=15*s.* 9.6*d.*

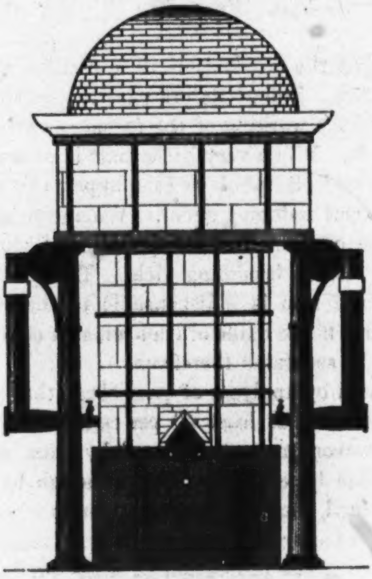
3. *Preparation of Lits de Fusion.*—The lits de fusion are usually composed as follow:—

Roasted ore	10,000 kilos.
Scrap iron	1,000 „
Limestone	1,600 „
Fluor spar	300 „

The above figures give the average quantities of flux in the furnace mixtures. They are, however, modified with the nature of the ore; the proportions of limestone and fluor spar varying most considerably. Two men are employed at this work, and receive 2f. each per day. They prepare a lit and a half in that time—15 tons consequently the cost is 0·27f. per ton of roasted ore, and 0·25f. per ton of crude ore.

4. *Smelting in Castillian Furnace.*—There are two of these at Pontgibaud, but it is seldom that more than one is in blast at a time. Fig. 78 represents a front elevation of one of these furnaces, which are constructed of cut lava, and are very inexpensive to build.

Fig. 78.



Castillian Furnace, Pontgibaud.
(*Front Elevation.*)

Their height from the slag-overflow *a* to the charging door is 5 feet, internal diameter 35 inches; diameter of tuyere 3 inches; pressure of wind about 4 inches of water. These furnaces are supplied with the blast by the nozzles *b*, of which there are three, connected with the main *c*; water tuyeres are not employed in the Pontgibaud furnaces.

The mode of charging is similar to that employed for other furnaces of the same kind.

The ore is distributed around the sides, the coke in the middle and against the breast. The furnace is kept constantly full, and particular care is taken not to let the flame appear above the charge; it being considered important to keep the top as dark as possible. A large breast pan, capable of holding 20 pigs at a time, is preferred, but it is sometimes difficult to maintain it of that size.

From 14 to 16 tons of ore are smelted in 24 hours, with a consumption of one ton of coke, or about 7 per cent. of fuel.

The quantity of lead obtained in the same time is from 100 to 120 pigs, or from 5 to 6 tons. From 7 to 10 per cent. of matt is also produced.

It is found that the production of a certain quantity of matt cannot be prevented. Indeed its presence is regarded as being in no way prejudicial to the working of the furnaces, unless it be formed in too great an excess. When very little matt is produced, the slags are generally rich in lead. Sulphide of iron appears to act on the silicate of lead as a powerful reducing agent. Whenever oxide of iron does not abound in the ores smelted, the presence of matt alone does not prevent the slags from becoming rich. The presence of a large quantity of oxide of iron is indispensable to the production of poor slags, since without it the oxide of lead remains combined with silicic acid, and cannot be separated therefrom.

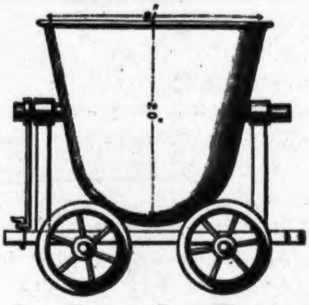
It has been found by analyses of poor slags that those most easily freed from lead contain at least 40 per cent. of oxide of iron. This base may be however, in part, replaced by lime, especially if fluor spar be at the same time added. Yet, although by this means poor slags can be obtained, they are never so poor as when oxide of iron is present in slight excess; the amount of lead volatilised is also considerably increased. The proportion of slags produced is from 65 to 70 per cent. of the ore smelted.

The slags formed under the most favourable circumstances do not contain less than 2 per cent. of lead. When they exceed 3 per cent. they are re-smelted in a furnace of similar construction to those already described, but situated in another part of the works.

The slags, as they flow from the furnace, are received into cast iron waggons, which, when full, are drawn away to the waste heaps on a small tramway constructed for that purpose. The waggons employed at Pontgibaud, of which Fig. 79 is a side view, Fig. 80 an end view, and Fig. 81 a plan, are very convenient in form, and but little liable to break. The depth of the pan is 20 inches, and its width 21 inches.

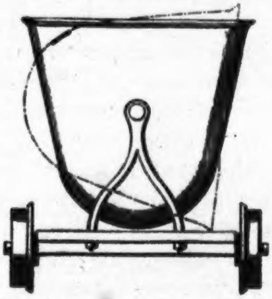
These have been in constant use for the last five years, and have scarcely ever needed repairs, except to the wheels and axles.

Fig. 79.



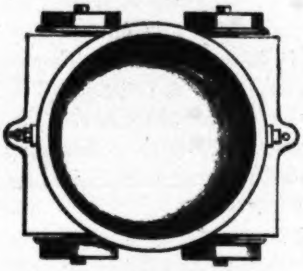
Side View of Slag Waggon.

Fig. 80.



End View of Slag Waggon.

Fig. 81.



Plan of Slag Waggon.

The men employed at this furnace per shift of 12 hours are:—

1 Foreman	paid		2·50 f.	per day.
1 Charger	„		2·20	„
3 Trammers	„	each	1·80	„

The costs per ton of *roasted* ore are as follow:—

Iron	110	kilos	at 95f.	10·45 f.
Limestone	160	„	at 20f.	3·20
Fluor spar	30	„	at 15f.	0·45
Coke	70	„	at 48f.	3·35
Labour				1·85
Tools and repairs				0·75
Sundries (including superintendence)				3·35
				F. 23·40=18*s.* 8·6*d.*

The cost per ton of *unroasted* ore is therefore 21·06f. or 16*s.* 10·1*d.*

The cylinders of the blowing machine are 1·34 m., or 52 inches in diameter.

Length of stroke, 52 inches.

Number of strokes per minute, 12.

The lead obtained from the Castillian furnace contains nearly all the silver originally present in the ores smelted, excepting a small proportion combined with matts, or retained in the slags. The usual assay of matt is from 15 to 20 per cent. of lead, and 400 to 500 grms., or from 12 oz. 17 dwt. to 16 oz. 2 dwt. of silver per ton; the average assay of the lead is about 3 kilos. = 96½ oz. per ton. The whole of the silver in the matt, and a portion of that in the slags, is recovered in the subsequent operations; but a small fractional part of that metal is nevertheless unavoidably lost. This loss amounts to 10 grms., or 6½ dwt. per ton of slag, or 0·568 per cent., according to assay, of the total quantity contained in the ores.

A certain amount of silver is also volatilised with the lead, but how large a proportion is, from this cause, entirely lost cannot be accurately ascertained. That it is, however, exceedingly small is probable, from the known properties of silver, and the small amount found in the fumes collected in the flues and condenser. The proportion of silver thus volatilised or mechanically carried off, and again recovered, in all the different processes, including cupellation, amounts to only 0·470 per cent. of the total quantity, according to assay, contained in the ores. The loss of lead in smelting ores in the blast furnace amounts to about 17 per cent. of the total quantity contained in them.

It has been ascertained that about 5 per cent. of all the lead is retained in the slags, and about 12 per cent. carried off in fumes. Two per cent. is, however, afterwards recovered from slags by re-smelting, and about 3½ per cent. from the fumes. The actual loss, therefore, in the operation is equal to 11¾ per cent. of the total quantity of lead contained in the ores.

It will be seen from the foregoing that the system of roasting and smelting at Pontgibaud has undergone very important alterations since the publication, in 1851, of Rivot and Zeppenfeld's "*Description des Gîtes Métallifères &c. de Pontgibaud*," which appears to be still regarded by many continental engineers as a description of what is being done at the present time. In proof of this it may be stated that in a lengthy paper on the Metallurgy of Lead, published in the "Revue Universelle des Mines," so recently as 1863, the engraving of a roasting furnace shown in Rivot and Zeppenfeld's work is reproduced as though it were still in use; whereas, in point of fact, it had been demolished ten years previously.

It is, however, in the treatment of the argentiferous lead obtained that the system now employed differs most essentially from that in use in 1851. At that period, the whole of the lead obtained from the blast furnace was immediately cupelled in a large German cupelling furnace, and the silver separated and refined in the usual way. The litharge produced was afterwards submitted to a complicated mechanical preparation in order to render it fit for sale; the whole, or very nearly the whole, of the lead thus converted into oxide being sold in that state. The small quantity of metallic lead sent to market was obtained, after great loss, by the reduction of a portion of the litharge, and was of very inferior quality. At the present time it would be extremely difficult to find a sale for so much litharge as the Pontgibaud works could now produce, as the yield of the mines has been enormously augmented. There is, however, never any difficulty in disposing of pig lead.

In the method employed at Pontgibaud since the works have been under the direction of Messrs. John Taylor and Sons, the object in view has been to convert the whole of the lead into metal of first-rate quality, and to obtain at the same time as complete a separation as possible of the silver: this is accomplished by means of Pattinson's process. The advantages of this method have been long appreciated both in England and on the Continent, and it is now generally adopted, either in combination with the old continental method, or with some modification of it suitable to the locality and the nature of the ores treated. The principal advantages of Pattinson's process, as compared with the old Pontgibaud method, are the following:—

1st.—Direct production of metallic lead.

2nd.—A greatly diminished loss of metal.

3rd.—A higher yield of silver.

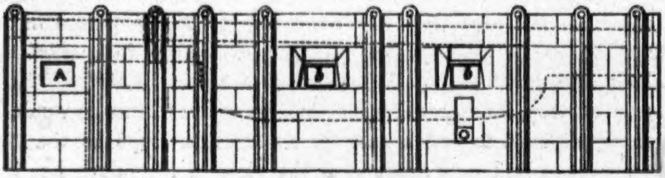
In the treatment of the argentiferous lead obtained from the blast furnace, four distinct operations are necessary, viz.:—

Improving or softening.
Crystallising.
Refining.
Reducing.

5. *Improving or Calcining.*—The whole of the work lead produced at Pontgibaud must be purified before it can be treated by Pattinson's

process, and this is done by exposing it at a low red heat to partial oxidation in a reverberatory furnace specially adapted for that purpose. The chief impurity contained in the lead is antimony; the others are sulphur, iron, arsenic, and copper. All are in relatively small proportion, but are still in sufficient quantity to render the lead *hard*. The accompanying drawings, Figs. 82, 83, and 84, show the arrangement and dimensions of the furnace employed.

FIG. 82.



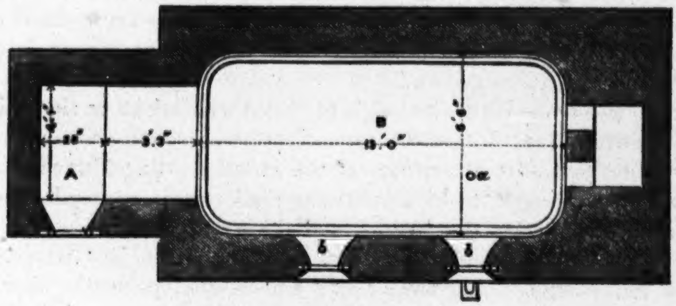
IMPROVING FURNACE.
(*Elevation.*)

Fig. 82 is an elevation; Fig. 83 a horizontal section at the level of the top of the pan; and Fig. 84 a vertical section through the tap hole. The fireplace A is separated from the pan B, by a bridge 3 feet 3 inches wide, and the furnace is provided with two doors *b*, through which the dross may be removed. In principle this resembles the ordinary softening furnace with its cast iron pan, but its greater size and solidity of construction render it much more economical than the furnaces usually employed for the purpose.

On reference to the drawing, it will be remarked that the pan is not only much larger than those commonly employed, but has also a rounded form; the object in giving it this shape being to diminish the tendency to crack, to which all square-sided pans are so liable. Another essential feature in the construction of these furnaces is to make them perfectly lead-tight, in case the iron should break. This is most effectually done by setting the pan on a bottom of well-beaten brasque two feet in thickness, resting on a solid foundation of masonry. The sides of the furnace must be either of thick iron plates or of large cut stones. In either case the space between them and the pan should be at least a foot wide, and well filled with hard beaten brasque.

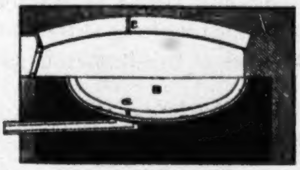
The lead is tapped from the pan through a small hole *a*, three-fourths of an inch in diameter, bored in the bottom of the pan, and communicating with a thick cast iron tube *b′*, fastened to it by means of stud-bolts screwed into about half the thickness of the metal. Before charging, the hole in the bottom is plugged by a long pointed bar passed through an opening *c*, Fig. 84, in the roof, corresponding with the tapping hole, and placed vertically over it. This bar will not generally stop the hole quite tight; especially after the furnace has been working a long time. The tube is, therefore, partly filled with bone-ash, firmly rammed in; a bar having been previously placed in the tube in such a way that the channel left, after its withdrawal, shall correspond with the hole in the pan. This horizontal

FIG. 83.



IMPROVING FURNACE.
(*Horizontal Section.*)

FIG. 84.



IMPROVING FURNACE.
(*Section through Tap Hole.*)

bar is even more necessary than the vertical one; the use of the latter being to take off the pressure of the lead in the pan, and regulate the flow of metal in tapping. The horizontal bar is put in and withdrawn by the aid of a sledge; in much the same way as, though more easily than, a tapping bar at the blast furnace.

Two of these furnaces have been erected at Pontgibaud; one for common work lead, the other for the hard lead reduced from calcined dross. The former has been almost constantly in use for upwards of five years; and although the pan is now cracked, the furnace is as tight and serviceable as ever.

Both improving pans were originally lined with bricks, as shown in the drawings, Figs. 83 and 84, to protect the iron from corrosion by the oxides formed on the surface of the metallic bath. This precaution is now thought to be unnecessary; all that is required being to avoid overheating the furnace, and the consequent fusion of the dross. It is also found that a dull red heat is the best temperature for calcining Pontgibaud lead; and at that point the oxides neither melt, nor exert any corrosive action on the pan, especially if a little lime be from time to time added. The usual charge of a pan without lining is about twenty tons. The brick lining diminishes its capacity by about one-fifth. A charge of twenty tons of common work lead requires sixty hours to become sufficiently soft for treatment by Pattinson's process, the whole time necessary for the operation, including charging and discharging, being three days.

Three men are employed in filling and emptying the pan, and are paid 2f. each per charge. The pigs of lead are introduced through one of the working doors by means of a long charging bar, and the charging is effected with great ease and rapidity. Except for charging and discharging, scarcely any labour is required, as the firing is attended to by one of the men working at the Castillian furnace.

An ordinary month's work at this furnace is as follows:—

Work Lead in. Tons.	Soft Lead out. Tons.	Percentage of soft Lead obtained.	Dross. Tons.	Coals consumed. Tons.	Lime used. kilos.
139·150	131·528	94·7	8·675	11·560	554

The small consumption of coal is partly due to the employment of cinders picked out from the imperfectly burnt ashes at the pots, and mixed with the coal; their cost per month being equal to a boy's wages, viz. 20f. per month. The cost of calcining one ton of blast furnace lead is—

Coal, 83 kilos. at 35f.	2·90f.
Cinders	0·15
Lime 7·5 kilos. at 24f.	0·18
Labour	0·45
Sundries	0·60
	F. 4·28 = 3s. 4·8d.

Cost on ton of ore 1·90f. = 1s. 6·2d.

The drosses resulting from the calcination of the work lead contain, besides the various impurities, a large proportion of lead, partially in the form of oxide, and partly in the metallic state. These drosses are subsequently treated in the reducing furnace.

The lead reduced therefrom is very impure and exceedingly hard, and, as it contains a considerable proportion of silver, 1,400 grammes, 45 oz., per ton, it is again calcined in the ordinary way. The hard lead obtained in the treatment of the lead cinder left as a residue in the reduction of litharge and pot dross is also calcined in the same furnace.

The hard lead contains a large amount of antimony and other impurities very difficult to separate, and consequently the process of softening in this case becomes long and costly. The average time required for calcining 20 tons of hard lead is about four weeks. The usual quantity treated per annum, with results and costs, is as follows:—

Hard Lead in. Tons.	Soft Lead out. Tons.	Percentage of soft Lead obtained.	Coal consumed. Tons.	Lime consumed. Tons.
148·845	96·978	65·1	99·900	5

Cost per ton of Hard Lead.

Coals, 665 kilos. at 35f.	23·27f.
Lime, 33½ „ at 24f.	0·80
Labour	0·43
	F. 24·50 = 19s. 7·2d.

Cost on ton of ore smelted 1·03f. = 9·6d.

The dross skimmed from this hard lead yields on reduction about 55 per cent. of a very hard lead, assaying 600 to 700 grammes, 19 oz. 5 dwt. to 22 oz. 10 dwt., of silver per ton; which is generally mixed with the ordinary hard lead and treated in the same way. In the statement of costs, all the hard lead, of every kind, is included. Occasionally, when a sufficient quantity has accumulated, it is worked separately. It then gives a dross which on being reduced yields an extremely hard lead. This, whenever it contains too little silver to pay the costs of extraction, is immediately disposed of for making certain alloys containing a large quantity of antimony. But this seldom happens, as it is only after repeated calcinations and reductions that a hard lead is obtained too poor in silver to pay for again working over. The proportion of poor hard lead produced at Pontgibaud is therefore exceedingly small.

6. *Crystallising.*—At Pontgibaud there are twelve pots in one set—eleven ten-ton pots, and one six-ton, or market pot. The system of working is by *thirds.* The ladles used are 20 inches in diameter, and 6 inches deep, and of the ordinary English pattern. The cranes employed are also of the usual form.

The average assay of the work lead is, as before stated, about 3,000 kilos., 96½ oz., of silver per ton. It is charged in the tenth pot.

The usual assays of the whole series of pots, tops and bottoms, are as follow:—

No. of Pot.	SILVER PER TON.								Remarks.
	Top.				Bottom.				
	grammes.	oz.	dwt.	gr.	grammes.	oz.	dwt.	gr.	
1	15		9	15	—	—			Poor Lead.
2	30		19	7	60	1	18	14	
3	60	1	18	14	120	3	17	4	
4	120	3	17	4	220	7	1	11	
5	230	7	7	21	400	12	17	5	
6	400	12	17	5	720	23	3	0	
7	700	22	10	3	1,200	38	11	16	
8	1,150	36	19	12	1,800	57	17	12	
9	1,850	59	9	16	2,900	93	4	21	
10	3,000	96	9	4	5,000	160	15	8	Charging Pot.
11	5,200	167	3	22	8,500	273	6	1	
12	8,500	273	6	1	16,000	514	9	1	Rich Lead.

In working the rich pot, the whole of the bottom is not ladled out; it is found to be more advantageous to ladle out the liquid only, and leave the crystals behind. When the ordinary quantity, or about two-thirds, of the lead has been turned into the eleventh pot, the remaining one-third consists of a mixture of crystallised and uncrystallised alloy. The latter, being much richer than the former, is separated as completely as possible; and this portion only, amounting to a little more than half the bottom, is sent to the refining furnace.

The work of removing the liquid is readily effected by means of what the French workmen call a *panier*, which is made of a piece of sheet iron, pierced with numerous small holes, turned up and fastened with rivets in the form of the frustum of an inverted cone. It is provided with two handles for putting it into and taking it out of the pot. By its use, a well is readily formed in the middle of the crystals, which facilitates the draining of the liquid metal from the surrounding mass, and allows of the lead being easily ladled into moulds. The right moment for introducing the panier, and ladling out the liquid, is easily determined by practice. When it is desired to take out much liquid metal, the pot is, on approaching the bottom, worked thin; but when the contrary is required, the work of stirring is continued until the crystals have become thick. In all cases, it is necessary to remove the whole of the liquid lead, otherwise the crystals left behind may be too rich.

The proportion of rich lead, viz. 18·2 per cent., cupelled at Pontgibaud is necessarily large, as the original lead is itself comparatively rich in silver. It is, therefore, very important to concentrate the rich alloy as much as possible before taking it to the refinery, where it is necessarily exposed to considerable loss from volatilisation.

Experiments made by Mr. Hutchison, with the object of enriching lead to a much higher degree than is usual, have demonstrated the impossibility of doing so beyond 2 per cent. of silver; and this degree of concentration is only attained after repeated and very careful crystallisations, and by at last drawing off a small quantity only of liquid alloy. When the converse of this operation is tried—*i.e.* when lead, enriched in the cupelling furnace so as to contain about 8 per cent. of silver, is fused in a pot, and cooled down in the ordinary way, the portions which first solidify are much richer than the liquid drained from them; the latter, after repeated drainings, invariably containing about 2½ per cent. It therefore appears that the remarkable property of crystallisation, discovered and utilised by Pattinson, arrives at its turning-point when the lead has acquired a richness of from 2 to 2½ per cent. of silver.*

In the ordinary way of working, it is remarked that in proportion as the concentration advances, it becomes more and more difficult to obtain the same degree of enrichment per pot or per ton of lead crystallised which is possible with poorer lead. An ordinary month's work at the pots may be tabulated as follows :—

---

* Mr. Hutchison remarks: "I have never seen any explanation of the reason for stopping the concentration at from 400 to 600 oz. per ton. We are generally told it would not be economical to push the concentration further; but this is not altogether true, as further concentration would certainly effect a saving of a part of the lead now volatilised during cupellation."

CHARGED.			OBTAINED.			
Calcined Lead. Tons.	Reduced Lead. Tons.	Total. Tons.	Market Lead. Tons.	Rich Lead. Tons.	Coals consumed. Tons.	Wages. f.
135·000	63·000	198·000	130·000	23·400	61·000	960

Four pairs of men are generally employed, whose time is divided into twelve-hour shifts. Four, and sometimes five, pots are worked by each pair per shift. The crystallisers are paid 0·60f. each per pot, and the stokers 2f. per shift.

The cost of crystallising per ton of market lead produced is as follows :—

Coal, 470 kilos. at 31f.	14·57f.
Wages	7·40
Wear and tear of pots	1·70
Tools and repairs	0·60
Sundries, including superintendence	4·00
	F. 28·27 = 22*s*. 7·4*d*.

Cost of crystallising per ton of ore, 12·39f. = 9*s*. 10·9*d*.

The loss of lead in the process of crystallisation proper is trifling, as the losses incidental to the process chiefly occur in the cupellation of rich lead and the reduction of drosses, and will be indicated under their respective headings. There is, however, a certain loss of silver in the market lead sold, since it invariably retains about 15 grammes, 9 dwt. 15 gr., of silver per ton; but this quantity amounts to only 0·533 per cent. of the total weight of silver contained in the ore according to assay.

7. *Refining.*—The ordinary English furnace is employed at Pontgibaud for the cupellation of rich lead. The operation is conducted in the usual way, except that the tests are deeper than those generally used. About four tons of rich lead are passed in twenty-four hours; but a certain quantity, amounting to one-fifth, is tapped out about every eight hours, or whenever the lead in the cupel has attained, by concentration, a richness of about 8 per cent. of silver. The bottoms of enriched lead thus accumulated are cupelled, as usual, at the end of the operation. About 15 tons of rich lead are usually worked at one time, and a plate of silver, weighing from 240 to 250 kilos. (7,720 to 8,042 oz.), is obtained. When cleaned, the silver is tapped at the

back of the test into moulds, and afterwards melted down in large black-lead crucibles. It is then cast into ingots of from 20 to 25 kilos. each, suitable for the market. These ingots contain 999 thousandths of silver.

The cost of cupellation per ton of rich lead is as follows:—

Coal, 290 kilos. at 35f.	10·15f.
Bone-ash 17 „ at 275f.	4·67
Pearlash 0·17 „ at 1,765f.	0·30
Repairs	0·30
Refiner's wages	8·23
Sundries, &c.	3·80
	F. 27·45 = 21*s.* 11·5*d.*

Cost of refining per ton of ore smelted, 2·27f. = 1*s.* 9·8*d.*

The loss of lead in refining is about 7 per cent. of the weight of lead worked, or 1·252 per cent. of the total work lead obtained from the blast furnace. The loss of silver volatilised with the lead vapours, although very minute, is probably greater during cupellation than in any of the other processes. Its amount cannot be determined with accuracy.

8. *Reducing.*—The furnace employed for reducing the litharge, pot dross, and calcined dross, is similar in form to the Welsh reverberatory furnace, but has only four working doors, two on the back side, and two on the fore side.

Its principal dimensions are as follow:—

	m.	ft.	in.
Height of furnace	1·75 =	5	8·8
„ from fire-bars to top of bridge	0·52 =	1	8·4
„ „ bridge to crown of arch	0·25 =	0	9·8
Width of fireplace	0·75 =	2	5·5
Length „	1·35 =	4	5·1
Width of bridge	0·60 =	1	11·6
Length of sole	3·70 =	12	1·6
Average width of do.	2·80 =	9	2·2

In the reduction of litharge and pot dross the operations are conducted in the usual way.

The average cost of reducing one ton of litharge and pot dross is—

Coal, 140 kilos at 35f.	4·90f.
Wages	1·15
Tools and repairs	0·40
Sundries, including superintendence	1·00
	F. 7·45 = 5*s.* 11·5*d.*

Cost on ton of ore, 1·73f. = 1*s.* 4·6*d.*

Nearly one-third the weight of market lead produced is skimmed off the different pots as dross, and passed through the reducing furnace. This large proportion of dross is due to the richness of the lead worked, and the consequent repeated crystallisations to which it must be submitted. The proportion of litharge reduced is nearly the same as that of the rich lead refined. The total weight of lead reduced from pot dross and litharge, and returned to the pots, is equal to nearly one-half the market lead made.

9. *Re-smelting rich Slags.*—The slags are smelted in a blast furnace of similar construction to those employed for smelting the ore. About 260 tons are smelted per month, and are derived principally from the ores actually worked during that time. Their average assay for last year was 3½ per cent. of lead.

The slags, after passing through the furnace, are thrown away, but still retain about 1¼ per cent. of lead. A very large proportion of the lead they contain is volatilised, but part of it is afterwards recovered in the flues, though in what proportion it is impossible to ascertain. Only 1¼ unit was obtained last year as pig lead (assay 750 grammes of silver per ton), and yet it was more than sufficient to pay the expenses of treatment; the fume collected in the flues of itself affords a fair profit. At present the slags are far too poor to make it advantageous to melt them over again, and it is probable that as the slags produced direct from the ore are gradually obtained poorer, this process will either be abandoned altogether, or only employed when the accumulation of rich slags may render it desirable.

The costs of smelting one ton of slag are the following:—

Coke, 100 kilos.	4·80f.
Smelter's wages	0·90
Tools and repairs	0·45
Cartage and breaking	0·77
	F. 6·92 = 5*s.* 6·4*d.*

Cost on ton of ore, 6·10f. = 4*s.* 10·6*d.*

SUPPLEMENTARY OPERATIONS.

a.—Roasting Matt.—The matt produced in the blast furnace is returned to the ore magazine, and mixed with ores in the "Lits de Grillage," as already explained; but before being mixed, it is ground in a mill to a coarse powder, and calcined in one of the reverberatory furnaces employed for roasting ore. A charge of matt weighs 2½ tons; each charge remains in the furnace double the time allowed for ore.

Every twelve hours a charge is withdrawn, and a fresh one introduced. The matt is worked on the bed next the fire, and as soon as a charge has become sufficiently hot, which takes place after about half an hour's brisk firing, it is, uninterruptedly, turned with paddles, during the whole time. Rapid oxidation of the metallic sulphides then commences; and as sufficient heat for calcination is thereby maintained, the fire in the grate is allowed to burn down, but not to go out, since in that case too much cold air would enter the furnace, and lower the temperature of the charge.

When the charge has been properly calcined it becomes almost black, contains very few lumps, and emits scarcely any sulphurous fumes. It is then drawn through the working doors, by means of iron rakes, and is thence removed to the ore magazine.

The cost of roasting one ton of matt is as follows:—

Coals, 150 kilos.	5·25f.
Labour	3·30
Grinding and cartage	1·00

F. 9·55 = 7*s.* 7·6*d.*

Cost of roasting matts on ton of ore, 1·05f. = 10*d.*

b.—Treatment of Calcined Dross.—The drosses skimmed from the work lead, and from the hard lead in the process of improving, are treated in the reducing furnace, and the residues smelted in the blast furnace. The object of this preliminary operation is to separate all the metallic lead by eliquation, and to reduce as much as possible the amount of the material to be treated in the Castillian furnace. It has been found that this is best effected by working it in small charges mixed with a certain proportion of fine coal. Five tons, divided into four charges of 1¼ ton each, are usually worked in twenty-four hours. The charge is turned and frequently paddled to facilitate the drainage of the lead from the mass, which acquires, after a short time, a rather pasty consistence. The cinder drawn out on the floor at the end of each operation is subsequently smelted in the manner described under the head of "*Treatment of Lead Cinder.*" About 56 per cent. of hard lead is thus obtained, with a consumption of 30 per cent. of coal.

The cost per ton of stuff is as follows:—

Coal, 305 kilos. at 35f.	10·68f.
Wages	2·22

F. 12·90 = 10*s.* 3·8*d.*

Cost on ton of ore, 0·68f. = 6·5*d.*

c.—Treatment of Lead Cinder.—These residues include the cinder from the reduction of the pot dross, calcined dross, and litharge. They are smelted in the blast furnace, with about twice their weight of common slag; no other fluxes are necessary. The coke employed as fuel, together with the carbonaceous matters contained in the cinder, effects its complete reduction. None but the usual precautions are necessary in smelting this cinder. Particular attention must, however, be paid to the charging, and to the proportion of slag added, otherwise there will be great risk of choking the furnace.

The quantity of cinder smelted per annum, with results, is as follows:—

Cinder smelted. Tons.	Lead. Per cent.	Slag employed. Tons.	Coke consumed. Tons.	Lead produced. Tons.
119·340	38·9	240·000	19·980	46·440

or 85·26 per cent. of total lead contained in stuff smelted.

The above quantity was smelted in twenty days, and cost for

Coke	959·00f.
Smelter's wages	404·00
Cartage	214·80
	F. 1,577·80

or per ton of cinder—

Coke, 16·7 per cent.	8·00
Wages	3·40
Cartage	1·80
	13·20 = 10*s.* 6·7*d.*

Cost on ton of ore smelted, 0·45f. = 4·3*d.*

d.—Treatment of Lead Fume.—Roasting.—Owing to the peculiar nature of the fumes collected in the flues at the Pontgibaud works, it has been found impossible to smelt them advantageously alone, as is done in various other establishments. Several methods have been tried and abandoned, but the best hitherto devised, and that which is now adopted, is to mix with the fume a certain proportion of silicious ore, and fuse them together in a common roasting furnace. The fused mass thus obtained, has the appearance of a clean black slag, and closely resembles ordinary roasted ore. It is subsequently smelted in the blast furnace in precisely the same manner as already described when treating of the smelting of ores.

The fume and ore are intimately mixed in the following proportions before being introduced into the furnace:—

Fume	60 parts,	assay 62·7	per cent. lead.
Ore	40 „	40·0	„ „
	100		

This mixture is divided into charges of 2½ tons each, and treated in exactly the same way as ore which has been roasted dead. It is charged *à la pelle* through the working doors of the furnace, upon the middle bed, where it is left undisturbed until the preceding charge has been melted and run out from the agglomerating bed. The charge on the middle bed is then immediately advanced towards the fire and piled up near the bridge; care being taken to move the mixture of ore and fume as gently as possible, otherwise considerable loss may occur from the fine particles being carried away by the draught. The whole of this charge having been advanced, and a new one introduced through the doors of the middle bed as before, the fire is urged so as to melt the mixture as quickly as possible, which is accomplished in from two to three hours. Five charges, or 12½ tons, can thus be passed in one furnace in twenty-four hours, the loss of weight in roasting being about 15 per cent.

The same number of men are employed as in roasting ore, but, instead of being paid by the day, they receive one franc each per charge. The consumption of coal is 17 per cent., and of lime 3 per cent. of the weight of stuff roasted.

The cost of roasting one ton of a mixture of fume and ore is as follows:—

Preparation of mixture	0·25f.
Coal, 170 kilos. at 35f.	5·95
Lime, 30 „ at 24f.	0·70
Labour	1·30
Cartage	1·00
	F. 9·20 = 7s. 4·3d.

Cost of roasting fume on ton of ore, 0·40f. = 3·8d.

Smelting.—The roasted stuff is broken and made into lits de fusion in much the same way as ordinary ore, but the proportion of fluxes added is much greater than usual.

The lits de fusion for fume and ore are composed as follow:—

Fume and ore	10,000 kilos.
Iron	1,200 „
Limestone	3,500 „
Fluor spar	300 „
	15,000 kilos.

The above quantity is usually smelted in twenty-four hours, with a consumption of 9 per cent. of coke.

The following are the costs of smelting one ton of the mixture of fume and ore in the blast furnace:—

Preparation of lits de fusion			0·40f.
Iron	120 kilos.	at 95f.	11·40
Limestone	350 „	at 20f.	7·00
Fluor spar	30 „	at 15f.	0·45
Coke	90 „	at 48f.	4·32
Labour			2·02
			F. 25·59 = 20s. 5·6d.

Cost per ton of unroasted mixture, 21·77f. = 17s. 4·9d.
Cost of smelting fume on ton of ore, 0·95f. = 9·1d.

The quantity of fume collected annually in the flues is about 154·900 tons, averaging 56·8 per cent. lead and 132 grammes, or 4 oz. 5 dwt. silver per ton. The richness of the fume varies considerably in different parts of the flue, but in general the percentage of lead increases, whilst the proportion of silver diminishes, with the distance from the furnaces. The lead obtained from fume by the process described, amounts to 78·57 per cent. of the quantity found by assay, or 3·67 per cent. of the total weight contained in the ores treated.

When it is considered how large a proportion of lead is lost by volatilisation in the form of fume by every known method of smelting lead ores, it evidently becomes a matter of importance to condense as large a proportion as possible of the metal thus driven off in order to reduce this loss to a minimum. Particular attention is now being devoted to this subject at Pontgibaud, and considerable improvements have been introduced with a view of recovering a larger proportion of fume than has been hitherto found possible.

Losses of Lead and Silver.—From 100 parts of lead contained in the ores treated, 85·75 are obtained either directly from them, or indirectly from slags or fumes, viz.:—

Direct from ores	80·04	per cent.
From fumes	3·67	„
„ slags	2·04	„
Total	85·75	„

In desilverising the lead thus obtained, a loss of 3·25 per cent. is experienced, the total weight of poor lead produced for sale being 82·50 per cent. of the quantity contained in the ores.

The loss in desilverising is distributed as follows :—

Refining	1·25	per cent.
Improving and reducing	2·00	„
Total	3·25	„

The loss of lead in the three principal divisions of the Pontgibaud process is therefore—

In roasting	2·50	per cent.
„ smelting	11·75	„
„ desilverising*	3·25	„
Total	17·50	

The percentage losses on the total quantity of silver contained in the ores are *—

In slags	0·568	per cent.
„ Market lead	0·533	„
Total	1·101	

The process now employed for the extraction of the silver is not only much superior to the old Pontgibaud method, but is even more perfect than the best methods of assaying at present known, since, in the large way, from 3½ to 4 per cent. more silver is obtained than the assays indicate.

Of 100 parts of silver produced—

98·82 are obtained direct from the ores.
0·64 „ „ from the slags.
0·54 „ „ „ fumes.
100·00

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* These losses are indicated by the assay of the slags and market lead, since the amount of silver annually obtained is, nevertheless, considerably in excess of the weight determined by assay.

The quantities of lead and silver sold from the Pontgibaud Smelting Works for the years ending June, 1865, and June, 1866, were as follow :—

			Value.
1865	Lead	1,391 tons	£61,582
	Silver	124,230 oz.	
1866	Lead	1,550 tons	69,420
	Silver	145,280 oz.	

Summary of Costs.

		Cost per Ton of Ore smelted.				Cost per Ton of Lead produced.				Cost per Kilo. of Silver produced.			
	f. c.	f. c.	£	s.	d.	f. c.	£	s.	d.	f. c.	£	s.	d.
Preparation of "Lits de Grillage"	0·25	21·45 =	0	17	1·9	48·94 =	1	19	1·8	16·78 =	0	13	5·0
Roasting Ore	19·75												
" Matt	1·05												
" Fume	0·40												
Preparation of "Lits de Fusion"	0·25	22·71 =	0	18	2·0	51,82 =	2	1	5·4	17·77 =	0	14	2·6
Smelting Ore	21·06												
" Fume	0·95												
" Lead Cinder	0·45												
Improving Ordinary Work Lead	1·90	2·93 =	0	2	4·1	6·69 =	0	5	4·2	2·29 =	0	1	10·0
Improving Hard Lead	1·03												
Crystallising		12·39 =	0	9	10·9	28·27 =	1	2	7·4	9·70 =	0	7	9·1
Refining		2·27 =	0	1	9·8	5·18 =	0	4	1·7	1·77 =	0	1	5·0
Reducing Pot Dross & Litharge	1·73	2·41 =	0	1	11·1	5·50 =	0	4	4·8	1·88 =	0	1	6·0
Reducing Calcined Dross	0·68												
Smelting Slag	...	6·10 =	0	4	10·6	13·92 =	0	11	1·6	4·77 =	0	3	9·8
		70·26 =	£3	16	2·4	160·32 =	£6	8	2·9	54·96 =	2	3	11·5

N.B.—General Expenses are not included in the above statement.

CHAPTER XXIII.

SMELTING SILVER ORES IN MEXICO.

ROASTING—FUSION—CUPELLATION—VASO—GALEME—S̃UFLA.

In Mexico, only the richer varieties of silver ore are smelted, by far the larger proportion being treated by the patio process; whilst in some few of the mining districts, as at Real del Monte, barrel amalgamation is extensively resorted to.

Roasting.—The ores destined for smelting are, at Zacatecas, and in some other localities, subjected to a process of roasting in heaps, which is effected by surrounding the mineral, broken into large lumps, with a layer of charcoal, retained in its place by an open wall of rough stone built in a circular form. The openings in this wall admit of the passage of the necessary amount of air, and the operation is completed in the course of twenty-four hours, with an expenditure of charcoal amounting to one-half the weight of the ores operated on.

At Nieves the ores are roasted in circular kilns four feet six inches in diameter, and of about the same height. These are formed of a hollow wall of adobes or sun-dried bricks, and are without a roof, the walls being so constructed that the area of the openings is nearly equal to that of the brickwork between them. Into each of these kilns are charged 2,000 lbs. of roughly-broken ore, intermixed with one-half its weight of dry wood; the operation requires a week for its completion; but this method of roasting can only be employed during the dry season.

Fusion.—The fusion of the roasted ore is effected in a small blast furnace, having the following dimensions: height to the charging hole 4 feet 6 inches, and depth from the front to the back wall 15 inches, width at top 11 inches, and at bottom only about 9 inches.

The aperture through which the fused materials make their escape is about 2½ inches in diameter, and is situated at the bottom of the furnace, whilst the copper nozzle through which the blast is supplied is placed about 9 inches above it. The breast, which is built of firestone, is taken down whenever the furnace becomes choked, or when it is found necessary to re-line the interior with a fresh coating of refractory clay; which is usually required at the expiration of a week from the time of first getting the apparatus into blast.

In front of the furnace, and below the aperture through which the various fused matters make their escape, is a breast pan 10 × 13 inches, in which they accumulate, and from which the lead and other fused matters are from time to time tapped into a basin, situated at a still lower level; whilst the slags are removed from the breast pan whenever any portion of them becomes sufficiently solid.

The blast is supplied by bellows, set in motion by cams, fixed in a horizontal axle connected, by toothed gearing, with a vertical shaft, to which a mule is attached. A couple of these furnaces are usually built side by side under a hood of masonry, open at the top for the escape of metallic fumes and the products of combustion. In front of each, is a series of bins containing the ores to be smelted, which are mixed in various proportions with the different fluxes employed. The fuel made use of is, in most instances, charcoal, prepared either from pine or green oak.

This furnace, which was first introduced into the country by the Spaniards, and appears to have undergone but little modification during the last three centuries, is that still almost universally employed in the different native establishments, although furnaces on the model of those used in the metallurgical works of the continent of Europe have replaced them in the larger haciendas, conducted under the superintendence of Europeans.

The materials employed as fluxes, or for the purpose of supplying the lead necessary for the extraction of silver, are *tequezquite*, or native carbonate of soda, collected during the dry season from the beds of certain lagunes; *greta*, or litharge, frequently very impure, and chiefly derived from the treatment of the lead ores of Mazapil and Zimapan; *temesquitate*, or the slags which float on the lead bath of the furnace known as the "ñufla," in which silver is extracted by a process of scorification; *crasas*, or the slags obtained during the progress of previous operations; and *fierros*, or the impure litharge, which floats on the surface of the lead bath during the earlier stages of the process of cupellation in the test furnace with a movable dome: this name is also applied to the matts obtained during the operation of smelting, which, together with cupel bottoms, and other materials containing lead, are added to the charge during succeeding operations.

The ore, after having been previously roasted, is placed in hutches before the furnaces, from which it is taken to be mixed with the various fluxes and furnace products constituting a charge.

At Sombrerete, where the rich ores are chiefly composed of quartz, iron pyrites, and the sulphides of silver, the furnace mixture was, in 1842, according to Duport, constituted as follows:—

Ore	37½ lbs.
Tequezquite	8 „
Temesquitate	8 „
Cupel bottoms	8 „
Fierros	12½ „
Litharge	50 „

The fusion of these ingredients produced a pig of lead weighing from fifty to fifty-eight pounds. The ores from La Cañada, which contain a larger proportion of lead, required a less amount of litharge, and somewhat less of the native carbonate of soda.

The campaign of a blast furnace begins on the Sunday evening, and extends to the morning of the following Sunday, at the expiration of which time the bad quality of the materials employed renders it necessary to suspend operations for the purpose of making repairs. After getting the apparatus into blast, 300 lbs. of lead ore are first passed through it, and of which the yield of metal is not calculated in the results obtained. As soon as the preliminary charge has been fused, the mixture of ore and fluxes is thrown into the furnace, with alternate charges of charcoal, as often as its contents descend sufficiently below the level of the charging hole.

The smelter, who has charge of two furnaces, takes care to keep the eye constantly open, and for this purpose frequently introduces an iron bar, with which he gently raises the slags and fuel which occupy the lower portion of the apparatus; he also attends to the proper regulation of the blast, and the proportion of fuel to be employed. When a sufficient amount of lead to form an ingot has accumulated in the breast pan, the blast is stopped, and the eye of the furnace is closed by a plug of clay; after which the dam, between the breast and receiving basins, is pierced, and the fused metal and matt are allowed to flow into the latter, when the eye of the furnace is again opened, and the operation continued until enough lead to yield another ingot has been produced.

As soon as the contents of the tapping pan have become sufficiently cooled, the matts are removed from the surface of the bath of lead, and at once passed through the furnace with the next charge, after which the metallic lead is ladled into a convenient mould. The foreman is assisted during this work by a man who charges the two furnaces, and by another who attends to the preparation of the mixture of ore and fluxes; he is relieved every twelve hours by another smelter and his assistants.

The quantity of charcoal consumed during the production of an ingot of metal varies from fifty to seventy-five pounds; and from eight to twelve are produced in the course of twenty-four hours, according to the nature and richness of the ore treated. When the minerals operated on contain a considerable amount of lead, each furnace will sometimes smelt as much as a ton of roasted ore in the course of twenty four hours, with a consumption of about 55 per cent. of charcoal; but such results can only be obtained under very favourable circumstances.

Cupellation.—The furnace usually employed for cupelling the argentiferous lead, resulting from the foregoing operation of smelting, is called a *Vaso*, and differs but slightly in its construction from many of those used for the same purpose on the continent of Europe. The hearth of the vaso is not, however, renewed after each working, but is employed for the cupellation of successive parcels of lead, until the test which, when new, is three feet in thickness, has become worn through. The bottom of the furnace, or *cendrada*, is composed of three parts of wood-ash and one of clay, well mixed together, and carefully beaten in.

In the largest description of cupelling furnace, the greatest diameter of the cendrada is 4 feet 6 inches, and of the smaller one, 4 feet; whilst the depth of the cavity for the reception of the metal is about 5 inches. The fireplace varies in its dimensions according to the nature of the fuel employed; requiring to be larger where the wood of the palm tree is made use of, than for furnaces in which pine is alone consumed. It is, however, in all cases situated very close to the edge of the cupel; but as this has itself a considerable diameter, it follows that towards the close of the operation the metal is at so great a distance from the fuel that the heat produced is far from being economically applied. The blast is furnished by bellows worked by two men, who relieve each other alternately; the bellows communicating with a movable tuyere capable of being depressed, in proportion as the bottom of the cupel itself becomes lowered. The blast thus applied drives before it the oxide of lead towards an opening in the opposite side of the hearth, and of which the level is lowered, in proportion as the depth of the metallic bath becomes less. These furnaces are not provided with chimneys, the products of combustion making their escape by a long low opening, situated opposite the fireplace, and on a level with the edge of the hearth, which also serves for the introduction of the ingots of metal to be cupelled. In order to prevent injury to the workmen from the action of volatilised oxide of lead, these furnaces, like those employed for smelting ores, are placed under large hoods of masonry, which effectually carry off the smoke and metallic vapours, so that the refiners are but little subject to diseases incidental to exposure to lead fumes. Although this apparatus is that still most commonly employed by Mexicans for the cupellation of argentiferous lead, the larger establishments now make use of the ordinary cupelling furnace fitted with a movable iron dome.

The *planchas* or ingots of argentiferous lead are taken from the smelting furnace to the vaso, which is capable of working off about two thousand pounds of lead in the course of twenty-four hours. The working of this furnace is conducted by a refiner and two workmen, who attend to the bellows, and are relieved every twelve hours by another foreman, and his two assistants who supply the blast. The fuel employed is usually pine wood split into small pieces, of which some six or eight hundred pounds weight are consumed in the course of twenty-four hours. In this furnace the process of refining is not generally carried so far as to allow the silver to brighten, but as soon as the metal remaining in the cupel begins to assume an iridescent appearance, the blast is arrested, and the fire withdrawn. The silver obtained by these means is, in the majority of cases, the property of individuals who sell it without its being assayed, and whose interest, consequently, is not to remove the whole of the lead; the resulting plates, therefore, usually contain from 970 to 985 thousandths of silver. In addition to lead, the silver obtained by this process generally contains a small amount of antimony, and requires to be refined, with an addition of pure lead, before it becomes sufficiently soft to admit of its conversion into coin.

The cost of smelting ores in Mexico is in all cases exceedingly high, but varies considerably in accordance with the nature of the power employed for supplying the blast. When, as is usually the case, mules are made use of for working the bellows, the expenses for power and manual labour are excessive, and at Zacatecas the costs under this head alone amounted, in 1842, to no less than $36 per ton, including the wages of the men employed for the cupellation of the resulting argentiferous lead: charcoal generally costs about 2¼ reals per arroba of 25 lbs., and these several expenses, added to the value of the litharge lost during each operation, made the total cost, at Zacatecas, of treating a ton of ore by this system, about $133, or 26*l.* 12*s.* per ton. The high price of smelting ores in Mexico is not, however, the only objection to the more general application of this process, since the loss of silver by the imperfect processes employed appears in all cases to be very considerable; varying, according to the nature of the ores and the skill of the workmen employed, from 15 to 25 per cent. on the assay value. The quantity of ore daily smelted at Zacatecas was estimated by Duport, in 1842, at from five to six tons only.

At Sombrerete, where the ores are generally rich, and where a larger proportion of them is, consequently, treated by smelting, the cost did not generally exceed about 13*l.* per ton; whilst at Nieves, which is at a more convenient distance from Mazapil, from whence the chief supply of litharge is obtained, and where the ores themselves contain a considerable amount of lead, the total expense of smelting a ton of mineral, together with that of refining the resulting work lead, was estimated at 6*l.* 6*s.* In some districts, in which water power is unusually abundant, the treatment of the ores is effected at an even less cost than the above; but such cases must, nevertheless, be regarded as altogether exceptional. The amount of silver obtained in Mexico by smelting, as compared with the total produce of the country, was estimated, in 1842, as being about one-tenth, but is now much less.

Galeme.—In some localities, a smaller, and more rudely-constructed apparatus than the vaso is employed for the cupellation of work lead obtained from the blast furnace. This arrangement, called a *galeme*, is constructed by marking out on a floor of well-beaten clay a space 2 feet 3 inches long, and 1 foot 8 inches wide, which is enclosed by a wall of stone, about 6 inches high, supporting a stone slab, or a large brick, forming the top, and covering the whole space. At one end is placed the fuel, consisting of pine wood, cut into small fragments, on which a current of air (supplied by bellows, worked by a lever) is directed, by means of an iron nozzle; whilst sundry small apertures at the other extremity allow of the escape of the products of combustion. Before charging the furnace, it is heated for some hours, in order to raise the temperature of the masonry to the proper point, and to so vitrify the hearth as to render it, to a great extent, impervious to litharge. An ingot of lead, weighing from forty to fifty pounds, is now introduced, and an aperture left in the side is partially stopped with clay, so as to allow the litharge to flow off; care being taken to lower its level in proportion as the surface of the metallic bath becomes depressed. The lead, thus placed in close proximity to the burning fuel, becomes rapidly converted into litharge, and the cupellation proceeds at the rate of about an ingot per hour, although the loss of that metal is considerable, and a deficit on the silver is the result. The galeme is the only means of cupellation employed in the mining district of Charcas, and is also made use of at Catorce, for refining the impure silver obtained by the treatment of silver ores in the cazo.

The Ñufla.—This furnace is probably the most primitive apparatus employed in any country for the extraction, by the dry way, of silver from its ores; and is in principle merely a scorifier, in which the cupellation of the remaining lead is subsequently effected.

The sulphides of silver sometimes occur in so large a proportion, with regard to the gangue with which they are associated, that if finely ground and thrown on a bath of metallic lead kept at a proper temperature, the silver combines with the lead; which, becoming partially oxidised, unites with the silicious and earthy impurities of the gangue, forming a slag, which may be readily removed from the surface of the metal before commencing the operation of cupellation. This apparatus, which is much employed by the native miners of Sombrerete, consists of a cupel formed of a mixture of ashes and clay, contained in a large coarse earthen dish; enclosed between walls of sun-dried bricks, and covered by a dome of somewhat refractory clay, in which numerous holes are pierced. The whole of the interior of this arrangement is filled with charcoal, on which a blast is thrown by means of a nozzle of baked clay; and, although the whole of the heat is applied from above the surface of the lead, it quickly melts, and the scorification of the ore, thrown on its surface, commences. When the workman considers that the bath of lead has, by the introduction of successive additions of ore, become sufficiently charged with silver, he cleans off the silicious slags, and commences the cupellation of the residual lead. For this purpose he makes an opening at the level of the metallic bath for the escape of litharge, and continues the operation, at the same time gradually lowering the level of the exit for litharge, until a button of silver alone remains in the bottom of the cupel.

CHAPTER XXIV.

ASSAY OF SILVER ORES AND BULLION.

ARGENTIFEROUS GALENA—ASSAY OF SILVER ORES PROPER—SCORIFICATION—CUPELLATION—ASSAY TABLE—ASSAY OF SILVER BULLION.

Assay of Silver Ores.—In the assay of ores of this metal, the object sought is first to obtain the silver present in the mineral in the form of an alloy with lead, which is subsequently passed to the muffle and cupelled, with proper precautions.

Argentiferous Galena.—The assay of argentiferous galena is, in this country, usually conducted in a wrought iron crucible. The vessel employed for this purpose is made of a piece of plate iron, of good quality, turned up in the form of a crucible, and carefully welded at the edges; the bottom is closed by a large iron rivet, securely welded to the sides, and the whole finished by the hammer on a properly-formed mould. To make an assay in a crucible of this description, it is first placed in the assay furnace, and heated to dull redness; and when it has become sufficiently hot, 400 grains of the pulverised ore, intimately mixed with its own weight of soda-ash, about 30 grains of pearlash, and from 8 to 10 grains of charcoal powder, or lamp-black, are introduced by means of a long copper scoop. With certain varieties of ore, the addition of a small quantity of common salt, or fluor spar, is found to be beneficial for the production of a thoroughly liquid slag; fluor spar being particularly advantageous in the case of highly silicious ores. On the top of this, is placed a thin layer of dried borax; and the crucible, which, for the introduction of the mixture, has been withdrawn from the fire, is at once replaced in the furnace. At first the contents of the pot boil somewhat violently, and therefore, in order to avoid loss, the crucible should be made of sufficient capacity to prevent any portion of the mixture being projected over its sides. At the expiration of from eight to ten minutes, the ingredients in the crucible will be observed to be in a state of tranquil fusion; and the pot must now be removed from the fire, and its contents briskly stirred by means of a small iron rod, flattened at the end in the form of a spatula. Any matters adhering to its sides are also scraped downward into the bottom of the pot, which is replaced in the furnace, and, after being closed with an earthen cover, is, during three or four minutes, heated to full redness.

The crucible is now seized, by a strong pair of bent tongs, on that part of its edge which is opposite to the lip; and, after being removed from the fire, its contents are rapidly poured into a cast iron mould, having internally the form of an ordinary egg cup. The sides of the pot are now carefully scraped down with the chisel-edged bar before referred to, and any adhering particles of lead and slag are obtained by sharply striking the edge of the crucible against the top of another cast iron mould, similar to that into which the assay was first poured. When sufficiently cold, the contents of the mould are readily removed by merely turning it over; and the metallic button, after being separated from the adhering slag, is carefully cleaned by means of a hard brush, and weighed, in order to determine the percentage yield of lead. When a metallic shot has been obtained in the second mould, it must be freed from adhering slag, and weighed with the larger button. The alloy thus obtained is cupelled, in order to determine the amount of silver which it contains.

Assay of Silver Ores Proper.—Ores of silver, in which that metal exists in the form either of oxide, sulphide, or chloride, in a gangue principally consisting of silica or of carbonate of lime, are usually fused with a mixture of litharge and carbonate of soda, to which a small quantity of finely-powdered charcoal is added; and by this means a button of alloy is obtained, which is subsequently treated by cupellation.

The proportion of litharge to be employed for this operation must be varied in accordance with the circumstances of the case, as the resulting button of alloy should not be too rich in silver, since, in that case, a portion might be lost in the slags; neither should it, on the other hand, be too poor, as the cupellation would then occupy a long time, and a loss through absorption be the result. In ordinary cases, where the silver principally exists in the form of chloride and sulphide, and the quantity operated on is 400 grains, a button of alloy weighing about 200 grains will be a convenient amount for cupellation: such a result may generally be obtained by the addition of 300 grains of litharge, 400 grains of carbonate of soda, 150 grains of borax, and from 7 to 8 grains of finely-powdered charcoal. The whole is to be well mixed, and introduced into an earthen crucible, of which it should not occupy more than one-half the capacity. Ordinary French pots are best adapted for making all assays in which large quantities of litharge are made use of, as they resist its action better than any of the pots usually made in this country.

The crucible is now placed in an assay furnace of the usual form, care being taken to withdraw it from the fire as soon as a thoroughly liquid and perfectly homogeneous slag has been attained; since the fused mixture would otherwise be liable to cut through the sides of the pot, and thus spoil the experiment. When it has sufficiently cooled, the crucible is broken and the metallic button obtained, which, after being properly cleaned, is passed to the cupel. When a great degree of accuracy is required, it is always best to break the pot, and extract the button of alloy, as above described; but when numerous assays have to be made on ores of nearly the same tenure, the assay is sometimes poured into an iron mould, and the crucible is employed for making other fusions. In this, and all similar cases, it is, of course, essential to ascertain, by previous experiment, the proportion of silver contained in the lead obtained by the reduction of the litharge, in order to obtain the necessary data for calculating the requisite deduction to be made from the results afforded by cupellation. When, however, very poor litharge is made use of, the resulting lead contains so small an amount of silver that, for some commercial purposes, its presence may be disregarded: generally speaking, however, the assayer, on the receipt of a fresh supply of that reagent, ascertains, by means of careful assays, the proportion of silver which it contains, and makes the necessary correction on each assay in which it is employed.

Argentiferous minerals containing a considerable amount of copper may be generally assayed by this process, since the amount of that metal which enters into combination with the lead produced is comparatively small, and the resulting button of alloy admits of being readily cupelled by the addition, when necessary, of metallic lead. When the mineral to be assayed contains a large proportion of metallic sulphides, the addition of charcoal, or any other reducing agent, becomes unnecessary, as litharge readily attacks all the simple and complex metallic sulphides, oxidising their constituents, with the exception of the precious metals, which form an alloy with the lead set free. The slags resulting from this operation contain the excess of litharge added, and the button of alloy produced is subjected to cupellation. The proportion of oxide of lead to be added to ores of this description varies in accordance with their composition, but it should in all cases be present in decided excess, since, should the sulphides not become completely decomposed, the whole of the silver will not be concentrated in the resulting button of alloy. For the successful assay of pure argentiferous iron pyrites, as many as 50 parts of litharge are required, whilst for mispickel, blende, copper pyrites, grey cobalt, and sulphide of antimony, from 15 to 20 times their weight may be employed.*

It must, however, be remembered that earthy and silicious gangues usually constitute a large proportion of the bulk of the ores operated on, and consequently these excessive amounts of litharge are, in practice, seldom requisite. One of the chief objections to this method of assay is the large amounts of lead that are produced for cupellation, since pure iron pyrites affords 8·50 parts of this metal, whilst sulphide of antimony, and grey copper ore, yield from six to seven parts.

This inconvenience may be obviated by effecting the partial oxidation of the sulphides, either by roasting or through the aid of nitre, by the skilful use of which a button of almost any required weight may be obtained. If this reagent be employed in excess it determines the oxidation of the various metallic and other oxidisable substances present, not always excepting silver itself. When, however, the mixture at the same time contains an excess of litharge, and nitre has not been added in sufficient quantity to effect the decomposition of the whole of the sulphides present, reaction takes place between the portion of sulphide undecomposed and the oxide of lead added. This gives rise to the formation of a button of metallic lead, which, combining with the silver, affords a button of alloy suitable for cupellation. The amount of nitre required to be employed for this purpose necessarily depends on the nature and richness of the ore operated on, but it must be borne in mind that 2·5 parts of nitrate of potash are sufficient to completely oxidise the constituents of iron pyrites, and that 1·5 and 0·70 parts, respectively, are, in the case of sulphide of antimony and galena, sufficient for this purpose.

When the ores contain a large proportion of sulphides, it is generally found most desirable to conduct the assay on the mineral after calcination. The roasting of the pulverised ore is best effected in a shallow scorifier, or earthen dish, into which a weighed quantity of the mineral to be operated on, (generally 400 grains) is introduced, and then carefully roasted in the muffle of a cupelling furnace. For this purpose the scorifier and its contents should be first placed in the mouth of the muffle, and kept constantly stirred with a thin bent iron rod; care being taken to commence the operation at a low temperature, since, from their great fusibility, such ores would be otherwise liable to agglutinate. As the calcination progresses, the scorifier may be gradually pushed further into the muffle, and thus subjected to successively increasing temperatures; as soon as sulphurous vapours are no longer evolved at a full red heat, the scorifier and its contents are withdrawn and allowed to cool. The ore, when sufficiently cold, is carefully removed from the earthen dish, and mixed, on a sheet of glazed paper, with the fluxes requisite for effecting its fusion, and the reduction of the quantity of lead necessary for cupellation. When the amount of mineral operated on is 400 grains, there should be added soda-ash 400 grains, borax 200 grains, litharge 400 grains, and charcoal 10 to 12 grains. The whole is now introduced into an earthen crucible, fused with the usual precautions, and the resulting button of lead passed to the cupel.

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* Berthier, Traité des Essais par la Voie Sèche, tome ii. p. 827.

Scorification.—This is a simple and convenient method of assaying ores containing the precious metals, when large quantities of metallic sulphides are present. The process consists in subjecting the finely-pulverised minerals, mixed with granulated lead and placed in a saucer-shaped earthen vessel or scorifier, to the action of a bright red heat in an ordinary assay muffle. A portion of the lead is thus converted into litharge, which, as fast as it is produced, combines with the various silicious and earthy constituents of the veinstone, forming slags, in which the other metallic oxides produced are taken up, whilst the silver and gold form an alloy with the lead remaining at the close of the operation. The scorifiers employed for this purpose should be made of well-baked close-grained fire-clay. It is necessary that they should be compact in their structure in order to resist the corrosive action of the litharge, and that they should be capable of withstanding sudden changes of temperature without breaking.

A number of these scorifiers corresponding to that of the assays to be made are selected, and into each are introduced 100 grains of powdered raw ore, intimately mixed with from five to eight times its weight of granulated lead, and a small quantity of dried borax. In all cases, however, the lead should be added in excess, as the resulting slags are thereby rendered more liquid. The granulated lead used for this purpose should, if possible, be almost entirely free from silver, but this is difficult to obtain, and when it cannot be procured it becomes necessary to estimate beforehand the amount of silver contained in the lead employed, and to make a corresponding deduction from the weight of the button afforded by cupellation.

The scorifiers, after being duly charged with the ore, lead, and borax, are taken to the furnace and introduced into the muffle, which has been previously brought to a full red heat. Their introduction at first considerably reduces the temperature of the furnace, and some pieces of charcoal should be placed in the opening of the muffle for the purpose of again raising the heat to the proper point. The muffle door is now closed, and in the course of a few minutes the lead enters into fusion, whilst white vapours are observed to rise from the assay, and the formation of litharge rapidly takes place. In proportion as the borax fuses, and the quantity of litharge increases, the contents of the scorifier soften; and as the temperature becomes more elevated, they enter into fusion, whilst the lead accumulates in the centre, in the form of a large metallic globule.

When the assays have reached a bright red heat, which is usually the case in from ten to fifteen minutes from the commencement of the operation, the stopper of the muffle is removed; and the current of air, which now enters, causes the oxidation of the lead to proceed more rapidly. In proportion as the litharge accumulates, the slag formed, by its combination with the earthy, silicious, and other matters contained in the ores, increases in quantity, and gradually extends itself over the whole surface of the lead. The door of the muffle is allowed to remain open about fifteen minutes, at the expiration of which time it is again closed, and the temperature is raised for about five minutes to full redness, for the double purpose of rendering the scoriæ as liquid as possible previously to pouring, and also to facilitate at the same time the reunion of any disseminated globules of metallic lead.

The scorifiers are now withdrawn by means of proper tongs, and their contents rapidly poured into moulds. When sufficiently cold, the buttons of lead are detached from the adhering slags by being hammered on a small anvil, and are then passed to the cupel. When this operation has been successfully conducted, the resulting buttons of alloy contain, practically, the whole of the precious metals present in the ore. It is, however, essential that the slags should be perfectly and uniformly liquid at the time of being poured into the moulds, for should they either be pasty or contain imperfectly fused lumps, a portion of the mineral will remain unacted on, and small metallic buttons may either be enclosed in the unfused part, or remain attached to the scorifier.

When, in spite of the temperature of the muffle and the other conditions of the process having been carefully attended to, the slags do not become sufficiently liquid, it is necessary to introduce an additional quantity of borax, and in some cases a little nitre may be added with advantage. Sometimes, although rarely, it is found necessary to stir the slags with an iron rod, for the purpose of dividing any lumps that may have been formed, and to incorporate them with the more liquid scoriæ.

This method of assay is applicable to all kinds of argentiferous and auriferous minerals, of moderate richness, containing large amounts of metallic sulphides; and, from its convenience and freedom from causes of error, it is very generally employed in establishments in which such ores are subjected to metallurgical treatment. When, however, very poor ores have to be operated on, the ordinary method of fusion with litharge is to be preferred, since larger quantities can thus be dealt with, and more accurate results consequently obtained.

Cupellation.—The cupellation of the buttons of argentiferous lead is conducted as described when treating of the assay of auriferous compounds, but as silver is, at high temperatures, more volatile than gold, the heat requires to be more carefully regulated than is necessary in the case of gold ores. In making all cupellations it is necessary to bear in mind that a cupel is only capable of absorbing about its own weight of litharge, and consequently a test should always be employed a little heavier than the button of alloy to be subjected to the operation.

The results obtained are also, to a certain extent, influenced by the temperature at which the cupellation has been conducted, and consequently all assays are liable to a small amount of error. If the muffle be too strongly heated, the silver becomes perfectly refined, but experiences a small amount of loss, through sublimation and the absorption of the cupel; whilst, on the contrary, when the temperature has not been sufficiently elevated, the button is liable to retain a small portion of lead. These two causes of error, existing at the same time in all cupellations, are found, in practice, to often nearly neutralise each other; although, in order to obtain uniform results from the same alloy, it is necessary to employ various minute precautions, both with regard to the temperature of the muffle and the condition of the cupels employed. The more important of these precautions have been specified when treating of the assay of gold ores, and will be found in detail in all works on the assay of minerals by the dry way, but a small amount of practical experience will be far more advantageous than any mere verbal instructions.

When, however, the amount of lead employed has been sufficient, the cupel is perfectly dry, is made of fine and well-prepared bone-ash, and the cupellation is conducted at a full cherry-red heat, the results obtained will, in almost all cases, be found of a satisfactory character. If the resulting buttons of silver be large they should not be abruptly withdrawn from the muffle, but be gradually drawn towards its mouth, since their sudden removal might cause them to "spirt," and a loss be thereby entailed. In the case of a very large button being obtained it is sometimes found advantageous to cover it, immediately after brightening, andbefore its removal from the muffle, by another cupel kept hot for that purpose.

When it has sufficiently cooled the metallic button is seized, laterally, between the jaws of a pair of strong pliers, and tightly squeezed, for the double purpose of loosening it from the cupel and detaching any adhering litharge. The button is then cleaned by the aid of a stiff brush, and weighed in a delicate assay balance. When, in addition to silver, the mineral under examination contains gold, the button obtained on the cupel is first weighed, and its weight noted; it is then flattened, dissolved in nitric acid, and the gold also weighed; the difference of the two weights thus obtained corresponds to the amount of silver present in the assay.

The weight of mineral employed for making an assay is, to a great extent, regulated by the amount of silver which the ores are supposed to contain. In Cornwall, for assays of argentiferous galena, the quantity operated on is often one ounce avoirdupois. In the mining districts of Nevada, two hundred grains are commonly made use of, and the contents of the crucible are often poured out into an iron mould. Scorifications are not easily conducted, in the common muffle furnace, on much above one hundred grains; but for assays of an ordinary silver ore by fusion, four hundred grains are conveniently employed.

For commercial purposes, the silver contained in any given mineral is, in this country, estimated in oz. dwt. and gr., one ton of 2,240 lbs. avoirdupois being taken as the standard of unity.

TABLE *showing the weight of Silver to the ton of Ore corresponding to the weight in grains and decimal parts of a grain, obtained from 400 grains of Mineral.**

If 400 grains of Ore give Fine Metal,	One ton of Ore will yield	If 400 grains of Ore give Fine Metal,	One ton of Ore will yield
gr.	oz. dwt. gr.	gr.	oz. dwt. gr.
0·001	0 1 15	0·200	16 6 16
0·002	0 3 6	0·300	24 10 0
0·003	0 4 21	0·400	32 13 8
0·004	0 6 12	0·500	40 16 16
0·005	0 8 4	0·600	49 0 0
0·006	0 9 19	0·700	57 3 8
0·007	0 11 10	0·800	65 6 16
0·008	0 13 1	0·900	73 10 0
0·009	0 14 16	1·000	81 13 8
0·010	0 16 8	2·000	163 6 16
0·020	1 12 16	3·000	245 0 0
0·030	2 9 0	4·000	326 13 8
0·040	3 5 8	5·000	408 6 16
0·050	4 1 16	6·000	490 0 0
0·060	4 18 0	7·000	571 13 8
0·070	5 14 8	8·000	653 6 16
0·080	6 10 16	9·000	735 0 0
0·090	7 7 0	10·000	816 13 8
0·100	8 3 8	20·000	1633 6 16

ASSAY OF SILVER BULLION.—The assay, by the dry way, of silver bullion comprehends three distinct operations. First, the accurate weighing of a given quantity of the alloy to be operated on; secondly, the cupellation, with due precautions, of this weighed portion with a proper amount of metallic lead; and, thirdly, the re-weighing of the pure silver thus obtained, when the loss of weight will correspond to the amount of alloy which has been removed. The furnace used for this purpose should be provided with the means of very accurately regulating the temperature; and is therefore generally constructed of a wrought iron shell, internally lined with firebrick, and has sliding doors arranged at each of its openings, by the aid of which the several apertures can, at the pleasure of the operator, be easily contracted or enlarged.

The quantity taken for examination is usually in this country a representative of the troy pound, which is subdivided into ounces and half-pennyweights, that being the lowest denomination to which assays of silver bullion are reported. It is evident that the assay pound may be represented by any actual weight chosen by the assayer, but from 10 grains to 1 gramme, about 15·44 grains, is the quantity most frequently employed by English assayers. An assay pound of 10 grains, with weights corresponding to the ounces and half-pennyweight, is very convenient for silver assays. The old assayers made their calculations in ounces, pennyweights, and grains, but it is more convenient to work by the decimal system, and, when trade estimations are required, to convert the decimal into trade expressions.

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* For the method of using this Table, see page 235.

By this method pure silver will be called 1,000, and English standard, which contains in the pound 11 oz. 2 dwt. of silver, with 18 dwt. of alloy, will be 925 in the thousand. Assayers therefore often speak of such an alloy as being 925 fine, and this expression is constantly employed in the different American mints. All excess of alloy over the English standard is, in this country, called "worseness," and Mexican dollars, which are composed of a mixture of 10 oz. 16½ dwt. of silver, with 1 oz. 3½ dwt. of alloy, would be reported as "worse" 5½ dwt. The true decimal expression of such a mixture would be 902·7, or worse 5 dwt. 8 gr.; but, as trade reports are only made to the half-pennyweight, it would be reported W. 5½ dwt.*

On the other hand, certain Indian rupees are composed of 950 parts of silver, united with 50 parts of alloy. This is equal to 11 oz. 8 dwt. of silver, combined with 12 dwt. of alloy, and hence their trade report would be "better" 6 dwt.

The first step to be taken in making an assay of silver bullion is to flatten out, in the form of a thin disc, the cutting taken from a bar. The edges of this are carefully cut off, and from the centre are prepared two assay pounds, which are wrapped in pieces of sheet lead, amounting to about one-half the weight necessary for effecting their cupellation. The amount of lead necessary for this purpose increases in proportion with the amount of alloy present in the specimen under examination, but even fine silver requires, according to Makins, three times its weight of that metal to ensure a satisfactory result. English standard silver will require six times its weight, and coarser varieties a proportionately larger amount. The estimation of the quantity of lead required is a matter to be determined by the experience of the assayer. It should, however, be as free as possible from silver, and the amount of that metal in the lead employed must be deducted from the results obtained.

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* Makins, "Manual of Metallurgy," p. 172.

The several weighed portions of the bars of bullion to be assayed, after being carefully wrapped in lead foil, are arranged in the compartments of a divided tray, having nearly the dimensions of the bottom of the muffle, so that the place of each cupel may correspond with its position in the furnace during the operation of cupellation. During the preparation of the different assays, the heat of the furnace must be got up, and a number of cupels placed in regular order in the muffle, so that they may become heated to the proper temperature. When the muffle and cupels have attained a uniform bright red colour, and the fuel in the furnace is in a compact state and free from cavities, the muffle door is closed by pieces of charcoal. This is done by commencing with large pieces, which are followed by a layer of smaller ones, and so on, until the orifice is about two-thirds built up.

The remainder of the lead necessary for cupellation is now introduced into the cupels by means of proper tongs, and, when it has become fused, the assays, wrapped in the necessary amount of lead foil, are placed in their respective cupels. When they have been all placed in their proper positions, the assayer completes the closing of the mouth by means of small twigs of charcoal, and, shortly afterwards, removes some of the upper pieces for the purpose of admitting a current of air, by which the starting of the cupellations is effected. This makes itself apparent by the metallic bath becoming uncovered, with the exception of certain small patches of oxide of lead, which circulate from the centre towards the edges. When the working has been thus fairly set up, the adjustment of the draught, by means of the various furnace openings, must be carefully attended to, in order to maintain a steady circulation from the centre towards the circumference, where the litharge, as fast as it is produced, is absorbed by the bone-ash of the cupel. During this operation little or no fume should be apparent, as the whole of the lead should, as far as possible, be converted into oxide, and absorbed by the cupel. In this way the globule of fused metal goes on diminishing in size, until finally, after the expiration of from twenty minutes to half an hour, the whole of the lead and base metals have become oxidised, and the phenomenon of brightening takes place.

When the operation has been skilfully conducted, the "going off" of the assays will commence with the front row, and gradually pass back, from row to row, to the last. When, on the contrary, the working off takes place irregularly, or commences at the far end of the muffle, it is an indication that the management of fire has been unskilfully conducted, and the resulting assays will turn out unsatisfactorily. It now only remains to remove the "mouth coal," or charcoal in the muffle door, which must, by means of a rake, be done quickly and completely; and the door of the muffle is fitted in its place. The whole of the openings in the furnace must be now closed, and the apparatus allowed to cool, during from one-half to three-quarters of an hour. At the expiration of this time the furnace is opened, and the assays taken out into a divided iron tray.

If the buttons obtained appear rounded on the top, or have a slight depression in the centre, and can be readily removed from the cupels, it is an evidence of the operation having been properly conducted; when, on the contrary, the buttons adhere firmly to the cupels, or throw out projections near the bottom, they are not fine; and if, in place of being convex and nearly spherical on their upper surfaces, they exhibit a flattened appearance, it shows that the cupellation has been effected with the addition of an insufficient amount of lead.

It now only remains to clean and weigh the assays, and to compare them with a well-ascertained standard of comparison consisting of weighed quantities of silver, of which the degree of fineness has been previously determined, and which have been cupelled in the same furnace, and at the same time, as the unknown alloys. It will be found that, in all cases, a loss has been experienced, which is partly due to volatilisation of silver, and partially to absorption by the cupel. The amount of this loss will depend on the conditions under which the cupellations have been effected, and will be in a great degree influenced by the temperature which has been employed. The standards, or proofs, subjected to the same conditions as the assays themselves, will necessarily experience a corresponding loss, and afford data for correcting the results indicated by the balance.

In establishments where large numbers of assays of silver bullion are daily made, the process of Gay-Lussac for estimating silver by means of a standard solution of chloride of sodium is extensively employed. As, however, this operation is rarely performed by the metallurgist, and comes rather within the province of the professional assayer, we refer for a description of it either to the work of the inventor of the process,* or to an article, chiefly translated therefrom, contained in Mitchell's "Manual of Practical Assaying."

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* "Instruction sur l'Essai des Matières d'Argent par la Voie Humide," par M. Gay-Lussac. Publiée par la Commission des Monnaies et Médailles. Paris. 1832.

APPENDIX.

A.—VOLATILITY OF GOLD AND SILVER.

It has been long known to Metallurgists, that both gold and silver are, under certain circumstances, to some extent volatile; but Napier has shown that when an alloy of these metals is kept in a state of fusion at a high temperature, they are very distinctly so. By collecting the metallic vapours escaping from a crucible containing from twenty-five to thirty lbs. of melted alloy (gold coin), the results obtained in two instances were respectively as follow :*—

Vapour collected. grains.	Pure Gold in Vapour. grains.
1.—4·80	4·50
2.—4·40	4·10

Deposits obtained from top, middle, and bottom of a chimney thirty-five feet in height, attached to a small furnace used for fusing an alloy of silver and copper (silver coin), gave the following results :—

	Top. per cent.	Middle. per cent.	Bottom. per cent.
Metallic Silver	3·30	9·19	29·95
Oxide of ,,	7·18	5·21	0·17

Deposits were also examined from the top, middle, and bottom of a chimney attached to a furnace in which an alloy of silver containing a considerable portion of gold was melted, with the following results :—

	Top.	Middle.	Bottom.
Metallic Silver	29·380	39·160	48·750
Oxide of ,,	1·982	3·140	0·032
Gold	2·120	2·640	4·250*

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* Quarterly Journal of the Chemical Society, 1857.

Makins has corroborated these results by experiments on deposits obtained from a flue attached to a muffle in which assays of an alloy of gold and silver had been made. 1,000 grains of this deposit gave—

Gold	0·087
Silver	0·763*

The lead carried off in the form of fume, during the metallurgical treatment of argentiferous galena, is invariably found to contain a certain proportion of silver, but the lead resulting from its re-treatment is always much less rich in silver than that obtained directly from the ores.

The following results, obtained from samples of fume taken at different points along the course of a flue leading to the stack of a lead-smelting establishment, in which the work lead contained, on an average, twenty-six ounces of silver per ton, show the relative volatility of the two metals when subjected to the heat of a Castillian furnace.†

From top of Flue.		Lead.	Silver per ton.			From bottom of Flue.		Lead.	Silver per ton.		
No.		per cent.	oz.	dwt.	gr.	No.		per cent.	oz.	dwt.	gr.
1	Near Castillian Furnace .	49	2	9	0	1	Near Castillian Furnace .	58	2	9	0
2	„ Reverb. „ .	20	3	5	8	2	„ Reverb. „ .	59	2	9	9
3	100 ft. in advance of No. 2	70	3	5	8	3	100 ft. in advance of No, 2	63	3	5	8
4	„ „ 3	44	3	5	8	4	„ „ 3	61	2	9	0
5	„ „ 4	48	2	17	0	5	„ „ 4	49	2	17	0
6	„ „ 5	47	3	5	8	6	„ „ 5	64	2	1	5
7	„ „ 6	46	3	5	8	7	„ „ 6	46	3	6	5
8	„ „ 7	40	2	1	5	8	„ „ 7	58	1	12	5
9	„ „ 8	42	3	5	8	9	„ „ 8	62	2	17	0
10	Throat of Refinery . . .	49	22	17	8	10	Foot of Chimney . . .	66	3	5	8

B.—MINERAL STATISTICS OF VICTORIA FOR THE YEAR 1866.‡

Number of Quartz Miners employed in the different districts :—

Ballarat	2,005
Beechworth	2,941
Sandhurst	4,118
Maryborough	2,292
Castlemaine	2,648
Ararat	874
Total	14,878

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* Quarterly Journal of the Chemical Society, 1860.

† The lead obtained from re-smelting slags is also, unless it occurs in the form of lead matt, relatively much poorer in silver than that obtained directly from the ores.

‡ Mineral Statistics of Victoria for the year 1866 : Melbourne.

Number of steam engines in operation during 1866 :—

Alluvial mining engines, 480 ; aggregate horse-power, 9,981.

Quartz mining engines, 510 ; aggregate horse-power, 9,231.

Number of stamp heads, 5,437.

Very full returns of the yield of gold from quartz veins have been obtained for the year 1866. It is not practicable to give an account of the yield from all the quartz put through the mills in the various districts, but accurate information has been received relative to 861,468$\frac{13}{20}$ tons, which gave 452,895 oz. 7 dwt ; or an average of 10 dwt. 16·2 gr. per ton :—

Mining districts.	Tons crushed.	Total produce. oz.	Total produce. dwt.	Yield per ton. dwt.	Yield per ton. gr.
Ballarat . . .	238,503¼	58,157	3	4	21
Beechworth . .	130,519$\frac{11}{20}$	118,495	19	18	3·7
Sandhurst . .	244,807¼	118,743	10	9	16·8
Maryborough .	79,552⅖	44,967	14	11	7·3
Castlemaine . .	124,374¼	85,662	3	13	18·5
Ararat	43,711	33,868	18	15	11·9
Totals	861,468$\frac{13}{20}$	459,895	7	10	16·2

The prices paid for treating quartz ranged from 4*s.* to 1*l.* 10*s.* per ton.

Quantity of gold exported during the year 1866 as returned by the Customs Department, 1,479,194¾ ounces.*

Returns show approximately the gold obtained from quartz veins and alluvial workings during the year 1866 to have been as follows :—

From quartz veins	521·017	ounces
,, alluvial workings.	958·177¾	,,
Total Gold exported . .	1,479·194¾	,,

C.—ST. JOHN D'EL REY MINING COMPANY.

The quantity of mineral raised from the mines, and the quantity of ore stamped during 1866, were as follow :—

	Tons.
Mineral raised	107·087
Ore stamped	60·685

The quantity of killas and other inferior mineral rejected at Morro Velho during 1866 was 45,931 tons, or 43 per cent. of the quantity raised.

The total produce of gold from the 23rd March, 1866, to the 23rd March, 1867, was 622,129 oitavas.

The net profit on the working of the mines from 23rd March, 1866, to 23rd March, 1867, was 109,407*l.* 15*s.* 8*d.*

The average yield of the ores at Morro Velho, for the year ending 31st December, 1866, was 9·780 oitavas per ton of ore stamped. By re-stamping with killas, and grinding (in arrastres), the refuse afforded in addition, 0·525, giving a total yield per ton of ore stamped during 1866, of 10·305 oitavas.

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* In addition to the above, 407,394 ounces of New Zealand gold were shipped from this colony during the year.

D.—DON PEDRO NORTH D'EL REY GOLD MINING COMPANY.

Stone treated and Gold returned for twelve months ending December 1866.

MAQUINE.		BAWDEN'S MINE.		DAWSON'S MINE.		TOTAL.	
Ore. tons.	Gold. oitavas.	Ore. tons.	Gold. oitavas.	Ore. tons.	Gold. oitavas.	Ore. tons.	Gold. oitavas.
12,779	70,348	4,787	11,754	106	85	17,672	82,137

	£	s.	d.
Value of 82,137 oitavas of gold at 8s. 7*d*.	35,250	9	3
Cost	20,162	8	5¾
Profit	£15,088	0	9¼

The returns made and profits realised from these mines during the first four months of the year 1867, have been as follow:—

	Gold produced. oitavas.	Profit. £	s.	d.
January	7,026	1,119	19	5
February	12,416	3,358	0	4
March	20,056	6,133	14	5
April	24,066	7,702	14	6
Total .	63,564	£18,314	8	8

E.—RETURNS MADE FROM THE ANGLO-ITALIAN GOLD MINES SINCE 1863.*

The Val Toppa Gold Mining Company.

	oz.	Value. £	s.	d.
1864	509·275	1,798	18	9
1865	574·575	2,032	14	5
1866	1,400·925	4,957	15	8
Total since 1863	2,484·775	£8,789	8	10

Val Toppa has yielded upon an average 1 oz. 5 dwt. of gold per ton of ore.

The Vallanzasca Gold Mining Company.

	oz.	Value. £	s.	d.
1864	381·650	1,105	14	10
1865	740·475	2,291	14	9
1866	1,746·925	4,966	16	3
Total yield of Gold since 1863	2,869·050	£8,364	5	10

Vallanzasca has yielded upon an average 15 dwt. of gold per ton of ore.

---

* Furnished by Mr. J. C. Goodman.

The Pestarena Gold Mining Company.

		oz.	Value. £	s.	d.
1866	January, Gold received . . .	776·350 . .	2,504	9	8
	August ,, ...	815·375 . .	2,638	0	8
	October ,, ...	556·650 . .	1,806	9	8
	December ,, ...	658·125 . .	2,176	17	9
	Total . .	2,806·500	£9,125	17	9
1867	January, Gold received . . .	265·850 . .	554	19	9
	May ,, . .	1,308·975 . .	4,293	17	8
	June ,, . .	1,154·950 . .	3,820	16	7
	Total . .	2,729·775	£8,669	14	0

	£	s.	d.
1866	9,125	17	9
1867	8,669	14	0
Total . .	£17,795	11	9

The Pestarena ore has yielded 2¼ oz. of gold per ton.

F.—RETURNS OF GOLD AND SILVER BULLION ASSAYED IN THE UNITED STATES DURING THE YEAR 1866, ON WHICH THE GOVERNMENT DUTY OF ONE-HALF PER CENT. WAS PAID.*

	Bullion assayed.
California	$49,020,250
Colorado	219,860
Idaho	535,105
Missouri	815
Nevada	15,272,546
New Jersey	13,688
New York	7,295,803
Oregon	4,785,231
Pennsylvania	4,044,218
Rhode Island	2,211
Utah	60,278
Washington	139,533
Total value	$81,389,538

---

* The duty is only paid on bullion assayed for the first time.

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INDEX.

PAGE

A.

ABERCROMBIE 120
Abrudbanya, Silver mines of . . . 258
Addition of salt to Torta 336
Africa, Gold districts of 27
,, Gold produced in 127
Aguacatal gold mines 88
Alabama, Gold in 36
Alexander Reef 122
Allemont silver mine 261
Allison Ranch vein 60
Alotepec, Silver mining district of . 311
Alps, Great Central Gold Mining Co. 118
,, The silver mines of 261
Alston and Weardale Workings . . 117
Altai Mountains, Silver mines of . 264
Amador County 55
,, district, Silver in 304
,, Quartz Mining Company . 56
,, silver mine 305
Amalgam, Native 248
,, Sodium 220
Amalgamated copper plate, Use of . 142
,, Plates 182
Amalgamation at Constante, Spain . 372
,, ,, Freiberg . . . 367
,, Barrel process . . . 364
,, Common pan . . . 393
,, Estufa 363
,, Hepburn and Peterson's pan . . . 399
,, of concentrated ores at Morro Velho . 218
,, of gold in pans . . 198
,, ,, the battery 177
,, of roasted ore in pans 407
,, Pan process . . . 390
,, separators 400
,, Varney's Pan . . . 395
,, Wheeler's 397
Amalgamator, Baux and Guiod's. . 196
America, Central gold districts of . 77
,, Colonies, North gold districts of 91
,, Gold fields of 29
,, North, Silver mines of . . 266
,, South, Gold districts of . 77
Analyses of auriferous pyrites . . . 198
,, native gold 3
Analysis of saltierra 326
Andreasberg, Silver mines near . . 261

PAGE

Angel's Camp 54
Anglo-Italian gold mines, Returns made since 1863 520
Anglo-Mexican Silver Mining Co. . 271
Annaberg, Silver mines of 260
Antimonial silver 248
Aorore, Gold fields of 123
Ararat 113
Argentière silver mine 262
Argentiferous and auriferous lead, Extraction of the precious metals from 448
,, galena, Assay of . . 504
,, gossans of Cerro de Pasco 313
,, lead, English system of treating . . . 453
Armens silver mine 253
Arquerite 248
Arrastre, Description of 168
,, *de marca* 329
,, Loss of mercury in . . . 333
Asia, Central and Southern, Gold districts of 26
,, Gold produced in 127
Assay of auriferous ores 226
,, gold bullion 240
,, Office, New York, gold coined 74
,, of pots at Pontgibaud . . . 486
,, silver ore, Comstock vein . 295
,, table 512
Asuncion silver mine 271
Atlantic or Appalachian gold fields . 29
Attwood's system of amalgamation 187
Auckland 122
Augustin's process for extracting silver 409
Auriferous quartz veins, Age of . . 10
,, sands, old and modern alluvial deposits . . 18
Austin silver mining district . . . 407
Australia, Discovery of gold in . . 102
,, Gold fields of 101
,, ,, produced in . . . 127
,, ,, washing in . . . 128
,, mode of working gold . . 110
,, power employed in working gold 113
,, scarcity of water for gold washing 109
,, South, Gold districts of . 121

PAGE
Australia, System of quartz crushing in 188
,, Vein mining in 165
Austrian Empire, Gold districts of 24
,, ,, ,, produced in 127
,, ,, produce of gold 26
Avonclift workings 117

B.

Badger mine 56
Bahu vein 82
Ballarat gold field 112
Banat of Temeswar 258
,, The silver mines of the 256
Bar diggings 129
Barrel amalgamation, Nevada 386
,, process of amalgamation 364
,, quartz 93
Basalt employed for mullers 329
Batea, Description of 167
Baux and Guiod's amalgamator 196
Beach mining 150
Beechworth gold mines 113
Bella Raquel Company 263
Bell's run 122
Bench diggings 129
Bendigo gold field 112, 117
Beschertglück silver mine 259
Big Oak Flat 52
Biscaina silver vein 270, 279
Bismuth silver 248
Blankets for washing 179
Blicken 451
Block riffles 145
Blower, The 132
Blue Lead 64
Bodie district 308
Bohemia, Gold districts of 25
,, Silver mines of 258
Boitza silver vein 258
Bolaños Silver Mining Company 271, 277
Bolivia, Gold districts of 90
,, ,, produced in 127
,, Silver mines of 315
Borax Lake 11
Bradford's Separator, Description of 194
Brazil, Gold fields of 79
,, ,, mining in 210
,, ,, produced in 127
,, ,, production of 86
,, results obtained from stamping mills 213
British Columbia, Gold districts of 98
,, ,, ,, produced in 127
,, Gold Mining Company 117
,, Possessions, Gold districts of 101
Bromide of silver 250
Bromyrite 250
Buckingham Gold Company 33
Buller River 125
Bullion assayed in United States, 1866 521
Bull veins 94
Buninyong Mining Company 117

PAGE
Buscones 274
Bute County 65
Butschum silver vein 258

C.

Cachoeira vein 82
Calaveras County 54
Calcination of worked lead 453
Calcining, Pontgibaud 481
California, Gold fields of 29
,, ,, produced in 127
,, ,, produce of 66
,, ,, washing 128
,, Quartz veins 47
,, Richness of gold deposits in 43
,, Table of the operations of the principal quartz mills in 1861 201
,, Vein mining in 165
Canada, Gold districts of 96
Canterbury, Province of 125
Cape Claim Company 65
Capellina 348
Carbonate of silver 250
Cardiganshire, Silver ores of 252
Cariboo 100
Carolina, North, Gold districts of 34
,, silver mine 261
,, South, Gold districts of 35
Castillian Furnace, Description of 442
,, ,, method of working 444
,, ,, Pontgibaud 477
,, ,, results obtained at Wildberg 447
Castlemaine district 113
Catherine Reef, United workings 117
Catorce Silver Mining Company 271
Cazalla silver mine 262
Cazo 358
Central American Mining Company, Silver mines of the 311
Cerro de Pasco silver mines 313
,, ,, town of 314
,, Proaña silver mine 278
Chalanches silver mine 261
Changes introduced at Pontgibaud since 1851 480
Charge of Cazo 361
,, continental cupelling furnace 450
,, Fondon 361
Charlotte mint, North Carolina 72
Chaudière Gold Mining Company 96
Chemical reactions of the patio 352
Chilian Mill, Description of 170
Chili, Gold produced in 127
,, produce of gold 90
,, Rock formations of 316
China, Gold districts of 26
Chloride of silver 249
Chlorination of ores at Freiberg 365
,, process, Gold 196

PAGE

Chontales Gold and Silver Mining Company . . . 77
Clausthal Furnace, Description of . 438
  ,, silver district of 261
Cleaning up sluices 144
  ,, stamps 183
Clear Creek 66
Cleft in the Rock workings . . . 118
Clogau vein 16
Clunes 114
Columbia, British, Gold districts of . 98
Common pan amalgamation . . . 393
Composition of ores at Freiberg . . 365
  ,, silver ores in Mexico 327
Comstock lode, names of mining claims 297
  ,, ores, Composition of . . 388
  ,, vein, Yield of silver from 299
Concave Buddle, Description of . . 191
Concentration of silver by crystallisation . 455
  ,, ,, in matts . 432
  ,, tailings in gold mining . . . 190
Confidence silver mine 304
Conrad Hill Mining Company . . 34
Constante, Amalgamation at . . . 372
  ,, cost of treating ores . . 384
  ,, melting and refining . . 383
  ,, method of working barrels 381
  ,, number of barrels employed 380
  ,, treatment of amalgam . 382
  ,, ,, residues . . 383
Consuelo mine 78
Consumido 353
Continental process of cupellation . 448
Copper plate amalgamated . . . 142
Cornwall, Gold in 14
  ,, Silver ores of 252
Coromandel 122
Corral mines 305
Corrugated quartz 93
Cortez district 305
Cosmopolitan workings 117
Cost of Barrel amalgamation at Ophir works 389
  ,, Barrel amalgamation at Real del Monte 385
  ,, crystallisation at Wildberg . 459
  ,, crystallising in England . . 466
  ,, ,, per ton of market lead, Pontgibaud 488
  ,, extracting silver by different processes 418
  ,, Patio amalgamation at Fresnillo 357
  ,, Patio amalgamation at Guanaxuato 355
  ,, Patio amalgamation in Nevada 358
  ,, refining in England . . . 465
  ,, roasting, Pontgibaud . . . 476
  ,, smelting, ,, . . . 477
  ,, treating gold quartz . . . 202
  ,, ,, ores, Constante . . 384

PAGE

Cost of treatment in Cazo 362
  ,, ,, Fondon . . . 362
  ,, working by Pan process . . 405
Coulterville 51
Coyoting 164
Cradle, Description of the 133
Crude smelting 428, 433
Crystallisation, French process . . 459
Crystalliser's wages 465
Crystallising, cost of, England . . 466
  ,, Pontgibaud 486
Csertesch silver vein 258
Culpeper mine 33
Cupel or test 461
Cupellation, Assaying 510
  ,, Continental process . . 448
  ,, Gold assaying . . . 230
  ,, in Mexico 500

D.

DAHLONEGA, Mint of 73
Daouria, Silver mines of 265
Daylesford 117
Defiance Company 117
Del Muerto silver vein 276
Denbighshire, Silver ores of . . . 252
Denver, Mint of 75
Description of Continental cupelling furnace 449
  ,, retort 404
De Soto silver mine 309
Devil's Gate district 310
Devon, Gold districts of 15
  ,, Silver ores of 252
Dimensions of mills, Constante . . 379
Discrasite 248
Distillation of amalgam at Freiberg . 371
Districts, Silver, lately discovered in Nevada 308
  ,, Silver-producing, of Peru. 318
Dognaczka silver mine 258
Dol-y-frwynog vein 16
Don Pedro North d'El Rey Company 84, 520
Dorn mine 35
Dorothea silver mine 261
Dry washing 163
Duero river, Gold-bearing sands of . 20

E.

EAST D'EL REY MINING COMPANY . 86
  ,, Indian Islands 27
Eastern Oregon 67
Éclair 451
El beneficio de cazo y cocimiento . . 322
El beneficio de hierro 322
El Dorado County 57
Embolite 250
English system of treating Argentiferous Lead 453
Esmeralda County 308
Estimation of gold in quartz without destroying specimen 235
Estufa amalgamation 363

PAGE

Encairite 249
Eureka canal 152
,, mine 56, 61
Europe, Gold produced in 127
,, Silver mines of 251
Expenses at the Kongsberg mines . 255
,, Real del Monte . . . 284
Extraction of gold from sulphides . 195
,, ,, silver ores at Guanaxuato 330
,, precious metals from argentiferous and auriferous lead . . . 443
,, precious metals by bath of melted lead . . 427

F.

Fahlbands 253
Fatzbay silver vein 258
Feistritz silver mine 262
Filtration of mercury 346
Fine grinding 329
,, stamping at Real del Monte . 351
Flat diggings 129
Fluxes, Materials employed as, Mexico 498
Fondon 360
France, Gold districts of 18
,, Silver mines of 262
Fraser river 98
Freehold Gold Mining Company . . 33
Freiberg silver mines 259
,, Amalgamation at 367
,, chlorination of ores . . . 365
,, composition ,, ... 365
,, Distillation of amalgam at . 371
,, Loss of mercury at . . . 370
,, Refining at 372
Freislebenite 250
French process of crystallisation . . 459
Fresnillo silver mine 278
Furnace, Castillian 477
,, ,, Description of . 442
,, Clausthal ,, . 438
,, Continental cupelling, Charge of 450
,, *Krumofen* 441
,, *Nuña* 503
,, Refining, Description of English 461
,, Roasting, Pontgibaud . . 472
Fusion of roasted ores in Mexico . . 497
Füzes silver veins 258

G.

Galeme 502
Galena, Argentiferous assay of . . 504
Galera 331
Gard, Gold districts of the . . . 19
[illegible]rdiner Gold Mining Company . . 33
[illegible]rnett and Mosely mines . . . 33
[illegible]udalcanal silver mine 262
Gem of the Sierras silver mine . . 309

PAGE

Geological position of silver 247, 251, 267
,, ,, ,, in Nevada 288
,, ,, ,, in Reese River mining region 301
Geology of Central America . . . 311
Georgia, Gold districts of 36
Germany ,, ,, ,, ... 24
Gold, extraction of, from sulphides . 195
,, Geological position of 4, 10, 30, 32, 40, 46, 57, 79, 90, 104, 125
,, hill mines 34
,, in alluvial deposits 12
,, large nuggets 4, 23, 29
,, mining; diagram of processes used in treating auriferous quartz 186
,, native, Analyses of 3
,, production of Africa . . . 127
,, ,, Asia 127
,, ,, Australia . . . 127
,, ,, AustrianEmpire26,127
,, ,, Bolivia . . . 127
,, ,, Brazil . . . 86, 127
,, ,, BritishColumbia 127
,, ,, California . . 66, 127
,, ,, Chili . . . 90, 127
,, ,, East Indies . . 27
,, ,, Europe . . . 127
,, ,, Merionethshire . . 16
,, ,, Mexico . . . 127
,, ,, Morro Velho . 83
,, ,, New Granada . 127
,, ,, NewSouthWales 120
,, ,, New Zealand . 127
,, ,, Nova Scotia . 96, 127
,, ,, Peru 127
,, ,, Russia . . 24, 127
,, ,, United States 76, 127
,, ,, Victoria . . . 119
,, ,, Wales . . . 17
,, refining 226
,, regions of the Old World . . 15
,, sources of supply 13
,, veins, Formation of 6, 12
,, ,, not impoverished in depth 7, 112
,, volatility of 11, 517
,, washing, comparative costs of various methods 161
,, washing in California and Australia 128
Goldlütte, Description of 209
Gongo Soco mine 1
Gould and Curry silver mine . . 295, 298
Granza 330
Grass Valley 60
Great Britain, Silver mines of . . 252
,, ,, and Ireland, Gold districts of 15
,, Comstock silver vein 287
,, ,, ,, ,, remarks on . 295
,, ,, ,, ,, principal ores of 293
,, Eastern silver vein 306

PAGE
Great Extended Workings . . . 117
Great Potosi mountain 315
Grinding and roasting ores, Constante 373
Ground sluice 149
Guanaxuato, extraction of gold from silver ores 330
" " mines 268, 270, 271
Gulch diggings 129

H.

Happy Valley 117
Hartz, Silver mines of the 261
Hayward's mine 9, 56
Hellefors silver mine 256
Hepburn and Peterson's Pan for amalgamation 399
Hessite 250
Hiendelaencina silver mines . . . 263
Hill diggings 129
Himmelfahrt silver mine 259
Himmelsfürst " 259
Hokitika River 124
Holland, Gold in 21
Hope Mining Company 118
Horizontal mills, Constante . . . 378
Horn Silver 249
Hot process of amalgamation . . . 358
Huelgöet silver mine 262
Hungarian Bowls, Description of . 204
Hungary, Gold districts of . . . 25
" " mining in 203
" Silver mines of 356
Hutchison's, Mr. observations on crystallisation 487
Hydraulic mining 154
" " cost of 160
" " results obtained by 160
" process 40
Hyposulphite of soda, lixiviation with 422

I.

Idaho, Gold districts of 67
Imperial Brazilian Mining Association 84
Improving, Pontgibaud 481
Incorporar 336
Independence mine 64
Inquartation, gold refining . . . 232
Iodide of silver 250
Iodyrite 250
Ione gold mine 62
Ireland, Gold districts of . . . 14, 17
Iron, volatility of 11
Isère, Gold districts of 18
Isle of Man, Silver ores of 252
Itacolumite 80
Italian's vein 57
Italy, Gold districts of 20

J.

Jamestown gold vein 53
Japan, Gold in 26
Joachimsthal silver mine 261
Johann-Georgenstadt silver mines . 260
Josephine mine 49

K.

Kallwang silver mine 262
Kernville 66
Kitzbühel silver mine 262
Koh-i-Noor workings 117
Kongens silver mines 253
Kongsberg district, Silver mines of . 253
Körösbanya silver vein 258
Kremnitz silver mine 257
" " ores of 427
Krumofen, Description of 441

L.

La Gardette, Silver mine of . . 261
Laidlaw's farm 93
Lama 331
Las Animas silver mine 307
Last Chance Hill, Silver ledges of . 308
Lavadero 342
Lead, cost of improving Pontgibaud . 485
Lemmond mine 35
Leviathan vein 57
Liberty Gold Mining Company . . 33
" silver mine 307
Limadura de plata 337
Lime, Employment of, in Patio . . 327
Liquation 466
Litharge 227
Lits de fusion, Preparation of, Pontgibaud 476
Lits de grillage 470
Lixiviating tub, Description of . . 411
Lixiviation 411
" with hyposulphite of soda 422
London and Virginia Gold Mining Company 33
Long and Muse's Mine 35
Loss by amalgamation of roasted ore in pans 408
" extracting silver by different processes 418
Losses of lead and silver, Pontgibaud 494
Loss of gold in amalgamation . . 189
" mercury at Freiberg 370
" " in arrastre . . . 333
" " in Patio process . 350
" silver " . . 349
" on working ores in barrels, Nevada 387
Lunenberg 95
Lyell River 125
Lyon County 310

M.

Machine employed for working tortas 341
Machinery employed at Real del Monte 283

PAGE
Madagascar, Gold in 28
Magenta aqueduct 153
Magistral, Preparation of 323
Maine Boy's tunnel 44
Malacates 273
Mammoth district 307
Management of Tortas at Zacatecas and Fresnillo 339
Manitowoc silver mine 309
Marienberg silver mines 260
Mariposa County 48
,, estate 5, 6, 47, 48
,, vein 50
Mariquita and New Granada Mining Company 87
Marlborough 124
Marmato mines 87
Marshall mine 33
Maryborough district 113
Massachusetts Hill 61
Massacre Bay 124
Matakitaki diggings 124
Materials necessary for reducing silver ores by Patio process 323
McAlpine's vein 51
McCulloch Gold Mining Company . 24
Mellado silver mine 274
Melting and refining, Constante . . 383
,, retorted gold 200
Merced Mining Company 50
Mercury, Employment of, in Patio . 326
,, Loss of, in Patio process . 350
Merionethshire, Gold in 15
Method of extracting silver ores of Comstock vein . . . 299
,, working Barrels, Constante 381
,, ,, Castillian furnace 444
Mexican Silver-Mining Companies . 271
Mexico, Cupellation in 500
,, Fusion of roasted ore in . 497
,, Gold districts of 77
,, Gold produced in 127
,, Important silver mines of . 267
,, Materials employed as fluxes in 498
,, Roasting ores in 497
,, Sale of silver ores in . . . 275
,, Silver mines of 266
,, ,, produced in 269
Miargyrite 250
Middle Hill silver veins 308
,, Yuba Canal Company . . . 152
Miners' Inch 152
Mines, silver, of Alotepec 311
Mining districts of Chili 316
Moldawa silver mines 258
Molino 329
Montana, Gold districts of 67
,, silver mine 309
Montezuma silver mine 308
Montgomeryshire, Silver ores of . . 252
Monthly amount of lead calcined, Pontgibaud 484
Monton, Weight of 335
Morro de Santa Anna 84
Morro Velho mine 80
,, Velho mine, Mode of occurrence of gold at 210
,, Production of gold at . . . 83
Moulding amalgam, Mexico . . . 347
Mountain of Alotepec 311
Mount Ophir vein 50
Murphy silver vein 306

N.

NAGYAGITE 1
Nagyag silver vein 258
Nagybanya, Silver mines of . . . 258
National aqueduct 153
Native amalgam 248
,, silver 247
Naumannite 249
Nelson 124
Neufang silver mine 261
Neusohl, silver mines of 257
Nevada, Barrel amalgamation in . . 386
,, County 59
,, First discovery of silver in . 296
,, giant silver vein 305
,, loss on working ores in Barrels 387
,, Quartz Mining Company . 62
,, results of Barrel amalgamation 385
,, settlers 392
,, Stamping ores in 391
New Granada, Gold fields of . . . 87
,, ,, ,, produced in . . 127
,, Orleans mint 71
,, South Wales, produce of Gold 120
,, Wombat Hill 117
,, York assay office 74
,, Zealand, geological remarks . . 125
,, ,, Gold districts of . . 122
,, ,, ,, exported . . . 125
,, ,, ,, produced in . . 127
,, ,, ,, mode of working 125
Nicaragua, Gold in 77
Nicolaiefsk silver mine 265
Norambagua gold mine 61
,, lode 166
North American colonies, Gold districts of . . . 91
,, ,, silver mines . . 266
,, Carolina, Gold districts of . . 34
,, River silver mine 304
,, Star Gold Mining Company . 61
,, ,, silver mine 303
Norway, Silver mines of 253
Nouveau Monde Gold Mining Company 50
Nova Scotia, Gold districts of . . 91
,, ,, produced in . . 127
,, Mining laws of . . . 95
,, produce of gold . . . 96
Ñufla furnace, The 503
Nuggets, Large 4, 29
Number of Barrels employed, Constante 380
Nye County 310

PAGE

O.

OBSERVATIONS on the crystallisation of lead by Mr. Hutchison 487
Offenbanya silver vein 258
Okanagan 100
Old Man vein 115
Oneida gold mine 56
Ophir, Mount 50
  ,, silver mine 295
  ,, works, Cost of Barrel amalgamation at 389
Oravitza, Silver mines of 258
Oregon 303
Ores from Comstock vein, Composition of 388
  ,, of silver 248
Otago 122, 124

P.

PACIFIC COAST gold fields 37
Palladium gold 1
Pan 131
  ,, process, Cost of working 405
  ,, ,, of amalgamation 390
  ,, Working in 401
Pans, Amalgamation of roasted ore in 407
Partido system of silver mining 279
Pasilla 337
Patio 334
  ,, amalgamation, cost at Fresnillo 357
  ,, ,, ,, Guanaxuato 355
  ,, Chemical reactions of 352
  ,, process for extracting silver 321
Pattinson's process 455
Pay dirt 128
Perdida 353
Peru, Gold fields of 88
  ,, ,, produced in 127
Philadelphia mint 69
Philipofsk silver mine 265
Phœnix Gold Mining Company 35
Piedmont, Gold districts of 20
Pine Tree mine 48
Placer County 58
  ,, diggings 40
  ,, mines 128
Pleiades, The, silver mine 306
Plumas County 65
Polybasite 250
Pontgibaud, Assay of pots at 486
  ,, Castillian furnace 477
  ,, changes introduced since 1851 480
  ,, cost of crystallising per ton of market lead 488
  ,, cost of improving lead 485
  ,, ,, roasting 476
  ,, ,, smelting 479
  ,, Crystallising at 486
  ,, Losses of lead and silver at 494
  ,, monthly amount of lead calcined 484
  ,, Reducing at 489

PAGE

Pontgibaud, refining 488
  ,, re-smelting slags 490
  ,, roasting matt 490
  ,, slag waggons 479
  ,, Smelting at 477
  ,, Summary of costs at 496
  ,, treatment of calcined dross 491
  ,, treatment of lead cinder 492
  ,, ,, fume 492
Porkhura 258
Porpezite 1
Porpylite 289
Port Phillip Gold Mining Company 114, 188
Portugal, Gold in 19
Potosi, Silver mines of 315
Poullaouen silver mine 262
Precious metals, Extraction of, by bath of Melted Lead 427
Precipitation of silver 423
Preparation of magistral 323
Pride of the Mountain silver mine 309
Prince of Wales mine 16
  ,, workings 117
Princeton vein 49
Process of extracting silver, Augustin's 409
  ,, ,, Von Patera's 419
  ,, ,, Ziervogel's 413
Produce and profit of the Real del Monte silver mines 286
Production of silver mines of Cerro de Pasco 315
  ,, ,, Chili 318
Profit and produce of Real del Monte silver mines 286
Proustite 250
Puddling box, Description of 138
Puy-de-Dôme 262
Pyrargyrite 249
Pyrites, auriferous 229
  ,, ,, Analyses of 198

Q.

QUARTZ MILLS, Table of the principal, in California, in 1861 201
  ,, mines 128
Quebra Panella vein 82
Queensland, Gold districts of 121

R.

RASPAR 332
Ravenswood district 305
Rayas silver mine 273
Real del Monte, Cost of Barrel amalgamation 385
  ,, fine stamping 351
  ,, machinery employed 283
  ,, results of Patio amalgamation 357
  ,, silver mines 278
  ,, Silver Mining Company 271
Reducing, Pontgibaud 489
Reduction works, Arrangement of 406

PAGE
Reduction works, Number of, Nevada 310
Reese River district 302
,, mining region 301
Refining at Freiberg 372
,, Pontgibaud 488
,, cost of in England 465
,, English process 461
,, furnace, English 462
,, gold 237
,, silver, Continental process 452
Relampago silver mine 263
Re-smelting slags, Pontgibaud 490
Results of Patio amalgamation at Real del Monte 357
Retort 405
Retorting amalgam, Mexico 347
,, description of process 198
Re-treatment of tailings at Morro Velho mines 216
Revenue silver mine 303
Reverberatory furnace, Charge for 437
,, ,, Description of 413
Rhine, Gold-bearing sands of the 18
Rhodium gold 1
River diggings 129
,, mining 149
Roasted ores, sifting and grinding, Constante 378
Roasting, Augustin's process 409
,, furnace, Pontgibaud, Description of 472
,, matt 490
,, Pontgibaud 472
,, silver ores, Mexico 351
,, Von Patera's process 419
,, with salt 410
,, Ziervogel's process 413
Rocker 133
,, Description of the 191
Rock sluices 147
Röhrerbüchel silver mine 262
Rohstein 435
Rossa Grande Gold Mining Company 85
Rough and Ready silver mine 305
,, stamping 328
Ruby silver 249
Russian Empire, Gold produced in 127
,, possessions, Gold districts of 21
Russia, production of gold 24

S.

SALA SILVER MINES 256
Sale of silver ores in Mexico 275
Salor River, Gold-bearing sands of the 20
Salt, Addition of, to torta 336
,, Roasting with 410
Saltierra 325
Salzburg, Gold district of 25
Samson silver mine 261
San Antonio, District mines of 307
,, ,, mine 78
,, Alvado silver vein 276
,, Bernabé ,, 276
,, Carlos silver mine 263
,, Cleménte ,, 276

PAGE
San Domingo silver mine 78
,, Francisco mint 70
,, Martin silver mines 276
,, Nicolas ,, 276
Sandhurst district 113
Santa Ana silver mine 319
,, Cecilia ,, 263
,, Rosa ,, 314
Sarah Sands 4
Savage silver mine 295, 304
Savoy, Gold in 20
Saxony, Silver mines of 258
Scarsdale workings 117
Scheidetrog 210
Schemnitz silver mine 256
,, Ores of 427
Schladming silver mine 262
Schneeberg silver mines 260
Scorification 508
Scotland, Gold in 17
Segersfors silver mine 256
Selbite 250
Selenide of silver 249
Semenofsk silver mines 265
Separation of gold and silver 237
Separators, amalgamation 400
Settlers, Nevada 392
Settling pits 183
Shaking tables, Description of 206
Sheba silver mine 309
Sierra Buttes Mining Company 63
,, County 63
Sifting and grinding roasted ores, Constante 378
Silicate of silver 250
Sil River, Gold-bearing sands of the 20
Silver, Antimonial 248
,, Bismuth 248
,, bullion, Assay of 512
,, Chloride of 249
,, combination of, with other metals 249
,, Commercial sources of 250
,, concentration by crystallisation 455
,, fusion with roasted lead ores without metallic iron 432
,, Geological position of 247, 251, 267
,, ,, in Nevada 288
,, ,, in Reese River mining region 301
,, glance 248
,, Hill, Mines in 308
,, mines of Central America 311
,, ,, Chili 316
,, ,, Norway 253
,, ,, the Old World 252
,, ,, Peru 313
,, ,, South America 313
,, mining districts of Europe 251
,, Native 247
,, ores 248
,, ,, Assay of 504
,, ,, Composition of, Mexico 327
,, ,, proper, Assay of 505

PAGE

Silver ores reduced by Real del Monte Company . . 285
" " Smelting of auriferous . 427
" " Treatment in Patio . . 321
" Peak district 307
" Peculiarities of Great Comstock vein 295
" Precipitation of 423
" produced by mines of Peru and Bolivia 316
" produced in Mexico . . . 268
" " Norway, from the Kongsberg mines 254
" produced in United Kingdom 252
" producing countries of the world, Yield of 320
" Production of, in Hungary, Transylvania, and the Banat 258
" Refining of, Continental process 452
" veins diminish in richness in depth, Mexico 277
" vein, Great Eastern . . . 306
" veins in Nye County . . . 310
" " Nevada Giant . . . 305
" " of Alotepec 311
" " Union district . . . 305
" volatility of 11
Slag waggons, Pontgibaud 479
Sleeping tables 208
Sluice, Description of the 139
" ground 149
" rock 147
" tail 147
" tunnel 149
" under-current 146
Smelting at Pontgibaud 468
" crude 428
" mixture, Pontgibaud . . 476
" of auriferous silver ores . 427
" with lead ores, *Bleiarbeit* . 434
Smoky Valley district 306
Sneath, Clay, & Co.'s mill . . . 62
Sodium amalgam 220
Soimanofsk mine 22
Sombrerete, Furnace mixture employed at 499
" Silver mines near . . 270
Sonneschmid 352
Soulsby's vein 52
Sources of supply of gold 13
South America, Gold fields of . . 79
" Australia, Gold districts of . 121
" Carolina, " . . . 35
Spain, Gold in 19
" Silver mines of 262
Spalling, Description of 211
Specific gravity of auriferous quartz . 236
Spitaler gangue 256
Spring Hill mines 56
St. John d'El Rey Mining Co. . 80, 519
Stamping mills, Description of . . 171
" in Australia . . . 111
" Brazil 211
" Hungary . . . 203

PAGE

Stamping ores, Nevada 391
" rough 328
Statistics, mineral, of Victoria . . 518
Stephanite 249
Sternbergite 249
Strakes employed at Morro Velho mine 214
Stromeyerite 249
Suerte silver mine 263
Sulphides, Extraction of gold from . 195
" of silver 249
" " Treatment of . 424
Summary of costs, Pontgibaud . . 496
Summerhill 120
Sutro Tunnel 300
Sweden, Silver mines of 256
Switzerland, Gold districts of . . . 21
Sylvanite 1
Szaszka silver mine 258

T.

TABLE, Assay 512
" for estimating the gold in quartz from its specific gravity . 235
" Mountain 46
" of yields of principal silver-producing countries 320
" showing amount of gold per ton of ore corresponding with a given weight per 600 grains 235
Tagus, Gold-bearing sands of the . . 20
Tahona 329
Tail sluice 147
Tajos de Panuco silver vein . . . 277
Taking sample from torta 338
Tallulah silver mine 309
Tamworth 120
Tasmania, Gold districts of . . . 121
Tcherepanofsk silver mine . . . 265
Tehuilotepec silver vein 270
Telegraph mill 52
Telluride of silver 250
Tennessee, Gold in 36
Tentadura 321
Teremakau 124
Testing quartz for gold 166
Test, or cupel 461
Thibet, Gold districts of 26
Tina Cargadora 343
Tipoani washings 90
Tlalpujahua Silver Mining Company 271
Tom, Description of the 137
Toplitza, Silver veins of 258
Torta, Washing up 342
" Weight of 344
Touchstone 244
Transylvania, Gold districts of . . 25
" Silver mines of . . . 256
Treading torta 340
Treatment of amalgam, Constante . 382
" calcined dross, Pontgibaud 491
" lead cinder 492
" " fume 492

PAGE
Treatment of residues, Constante . 388
,, silver ores in Bolivia, previous to Spanish conquest 315
,, sulphide of silver . . 424
Trinidad mine 78
Try-again Gold Mining Company . 118
Tub, Lixiviating 411
Tunnel sluice 149
Tuolumne County 51
Turon 120
Twin River district, veins of . . . 306
Tyrol, Gold in 25
,, Silver mines of the 262

U.

Under-current sluice 146
Union district 305
United Extended Band of Hope workings 117
,, Mexican Silver Mining Co. . 271
,, States, bullion assayed, in 1866 521
,, ,, First gold coined in . 30
,, ,, Gold coined at mints. 76
,, ,, ,, fields of the 29, 66
,, ,, ,, production of 76, 127
Utah, Gold in 67

V.

Valenciana silver mine 271
Vallanzasca Gold Mining Company . 520
Val Toppa 520
Vanderbilt silver vein 306
Vandorburg Mining Company . . 35
Varney's pan, amalgamation . . . 395
Vascongada silver mine 263
Vaso 500
Vein mining in Australia—California 165
,, ,, Brazil—Hungary . . 203
Veneros 90
Verdad de los Artistas silver mine . 263
Veta de Colquirirca ,, . . 314
,, Pariarirca ,, . . 314
,, Grande silver vein . . . 267, 276
,, Madre 267, 271
Victoria, Act for management of gold fields 118
,, Geological remarks on gold fields of 104
,, Gold exported from . . . 119
,, ,, fields of 103
Vigra and Clogau 16
Villa Guttier silver mine 263
Virginia, Gold districts of 32
Vitreous sulphide of silver . . . 248
Volatility of gold and silver . . 11, 517
Volcano gold mines 57
Von Patera's process 419
Vöröspatak, silver vein 258
Vulkoj ,, ... 258

W. PAGE

Waipori mines 125
Wakamarina 124
Walchern silver mine 262
Wales, Gold produced in 16
Waller Gold Mining Company . . 33
Wangapeka 125
Washing up torta, Guanaxuato . . 342
,, Zacatecas . . . 345
,, Fresnillo . . . 345
Washington district 305
,, territory 67
Washoe County 310
Water, amount required, cost of . . 161
,, Extraction of, from mines in Mexico 273
,, from the Comstock silver vein, Analysis of, &c. . . 293
,, supply 151
Weighing 234
Weight of a monton 335
,, torta 344
Welcome vein 115
West's furnace 373
Wet processes, Obtaining silver by . 409
Wheeler's pan, amalgamation . . . 397
Whitehall mine 33
Whitlatch Union silver mine . . . 304
Whitman gold vein 57
Wicklow, Gold in 17
Wildberg, Cost of crystallisation at . 459
,, results obtained from Castillian furnace . . . 447
Wood consumed by the Virginia Mining Companies 298
Wood's Point workings 118
Working in pans 401
,, torta "hot" 339
Work lead, Calcination of 453

X.

Xanthocone 250

Y.

Yauricocha 314
Yield of the Potosi silver mines . . 316
,, principal silver-producing countries 320
Yosemite silver mine 309
Yuba County 65
Yungas, Gold washings of 90

Z.

Zacatecas, production of silver . . 278
,, Silver mines of . . . 275
Zalathna smelting works 258
Ziervogel's process 413
Zigzag riffles 145
Zimapan silver mines 270
Zméof silver mine 264

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